

A DETAILED REVIEW AND EVALUATION
OF PRE-DRAINAGE TECHNIQUES AND THEIR
APPLICABILITY TO CANADIAN COAL MINES

FINAL REPORT

BY

ASSOCIATED MINING CONSULTANTS LTD.

WITH

NOWSCO WELL SERVICE LTD.

Work on this project was conducted under the auspices of the Canada Centre
for Mineral and Energy Technology; Energy, Mines and Resources, Canada.

277 PP

SSC File No: 03SQ.23440-7-9016
SSC Contract Serial No: 23440-7-9016/01-SQ
CANMET Scientific Authority: Dr. B. Konda

PERMIT TO PRACTICE
ASSOCIATED MINING CONSULTANTS LTD.

Signature

Alan L. Brown

Date JUN 15 1988

PERMIT NUMBER: P 2361

The Association of Professional Engineers,
Geologists and Geophysicists of Alberta

AR-124
MARCH 1988

MICROMEDIA

AMCL



**Associated
Mining Consultants Ltd.**

1400, 910 - 7 Avenue S.W., Calgary, Alberta T2P 3N8
Tel: (403) 264-9496 Telex: 03 821201

June 15, 1988
File: 0302-PC03
AR-124

Cape Breton Coal Research Laboratory
CANMET, Energy, Mines and Resources
210 George Street
SYDNEY, Nova Scotia
BIP 1J3

Attention: Dr. B. Konda
Scientific Authority

Dear Sirs:

Re: A Detailed Review And Evaluation Of Pre-Drainage
Techniques And Their Applicability to Canadian Coal Mines
SSC File No: 03SQ.23440-7-9016
SSC Contract Serial No. 23440-7-9016/01-SQ

We take pleasure in submitting our final report on Pre-Drainage in Coal Mines.
We would like to thank CANMET for sponsoring this study and for the advice and
guidance during its execution.

Respectfully submitted,

ASSOCIATED MINING CONSULTANTS LTD.

Hans U. Bielenstein, P.Geol.
Project Manager

HUB/tlm/88-5

Enclosure



AR-124

PRE-DRAINAGE OF COAL MINES

TABLE OF CONTENTS

Executive Summary

Sommaire

SECTION 1	INTRODUCTION
SECTION 2	PRE-DRAINAGE CONCEPTS
SECTION 3	CURRENT APPLICATION (CASE HISTORIES)
SECTION 4	IN-SEAM DRILLING
SECTION 5	IN-SEAM GAS FLOW STIMULATION
SECTION 6	DISCUSSIONS AND CONCLUSIONS
SECTION 7	CAPE BRETON TEST PROGRAM
SECTION 8	REFERENCES/SELECTED BIBLIOGRAPHY
APPENDIX A	DETERMINATION OF GAS CONTENT OF COAL SEAMS (LAMA AND BARTOSIEWICZ, 1982)
APPENDIX B	FACTORS AFFECTING GAS RELEASE FROM THE WORKING SEAM (HAYES, 1982)
APPENDIX C	RESERVOIR ENGINEERING IN COAL SEAMS: PART 1: THE PHYSICAL PROCESS OF GAS STORAGE AND MOVEMENT IN COAL SEAMS PART 2: OBSERVATIONS OF GAS MOVEMENT IN COAL SEAMS (GRAY, 1987)
APPENDIX D	EFFECTS IF STIMULATION TREATMENTS ON COALBEDS AND SURROUNDING STRATA (DIAMOND AND OYLER, 1987)

EXECUTIVE SUMMARY

Methane gas has always been a problem in underground mining. One approach to reduce the total methane problem is to pre-drain the gas from the seam prior to mining. A world wide review has shown that in Europe and Australia, gas pre-drainage has been used to overcome ventilation and outburst problems. In the United States, the systems have been developed primarily to obtain a saleable fuel with the additional benefit to ventilation when the coal is mined. In both cases, the final drainage designs were site specific and depended on a clear understanding of local coal seam conditions. This study deals specifically with the drainage of gas prior to mining from the coal seam only.

Coal as a gas reservoir is quite distinct from other reservoirs (e.g. sandstone, limestone) since only 10% of the gas occurs in the pore space, the remainder is adsorbed on the coal surfaces. In order to release this methane, a pressure gradient must be established. Coal has a low elastic modulus and high Poisson's ratio; the permeability of the coal reservoir, therefore, is very sensitive to pressure changes. Experience has shown that the presence of water severely inhibits reservoir permeability.

Methane pre-drainage requires drillholes into the block to be drained. Drilling technology has developed the capability of producing holes up to 1000 m long, but the directional control of long holes has not yet been perfected. Shorter holes up to 200 m in length are preferred for better directional control and spacing.

Hydrofracturing of coal reservoirs has been carried out in numerous areas to increase the gas delivery to the drillhole, but to date, only from surface and with partial success.

The offshore mines in the Sydney Coalfield are working in two gassy seams. Modern retreat longwalls can produce large tonnages, but methane control on the face is by ventilation (dilution) alone. This system may limit the face production as mining progresses to greater depths. Any methane captured from the seam before longwall extraction will help to achieve high production.

A phased research program is recommended for the Sydney Coalfield to assess the feasibility of methane pre-drainage. The initial phase would consist of drilling, testing coal core and in-situ testing for reservoir assessment. The second stage would provide further drilling, sample testing and reservoir monitoring. These holes would be used during a Phase 3 program when the reservoir would be stimulated through hydrofracturing from underground workings.

It is recommended that the initial trial be carried out in the Phalen Colliery because the retreat longwall development allows access to the longwall panels prior to extraction, current workings are relatively close to the portals and the Phalen Seam may be destressed by overmining in the Harbour Seam.

SOMMAIRE

Le méthane a toujours été un problème dans les mines souterraines. L'une des manières d'aborder le problème du méthane dans son ensemble serait d'appliquer le procédé du pré-drainage du gaz de la couche de charbon avant de commencer l'exploitation minière. Une étude de ce qui se passe ailleurs dans le monde a démontré qu'en Europe et en Australie le pré-drainage du méthane a servi à résoudre les problèmes de ventilation et d'explosion. Aux États-Unis, des systèmes ont été développés dans le but surtout d'obtenir un combustible vendable. Ces systèmes ont eu comme effet secondaire d'améliorer la ventilation dans les exploitations charbonnières. Dans chacun des deux cas, les conceptions finales du drainage étaient adaptées à leur emplacement et dépendaient d'une compréhension claire des conditions de la couche locale de charbon. Le présent rapport couvre spécifiquement le drainage du méthane avant le début de l'exploitation à partir de la couche de charbon seulement.

Le charbon en tant que roche-réservoir de gaz se distingue des autres roches-réservoirs (ex. grès, calcaire) en ce que seulement 10% du gaz se trouve dans l'espace poreux, le reste est absorbé sur les surfaces du charbon. Si l'on veut laisser échapper le méthane, un gradient de pression doit être établi. Le charbon possède un faible module d'élasticité et un coefficient de Poisson élevé; la perméabilité des roches-réservoirs est donc très sensible aux changements de pression. L'expérience démontre que la présence de l'eau diminue sévèrement la perméabilité des roches-réservoirs.

Le pré-drainage du méthane nécessite des trous de forage dans les blocs à drainer. La technologie du forage est maintenant capable de produire des trous allant jusqu'à 1000m de longueur, cependant le contrôle des trous directionnels d'une telle longueur n'a pas encore été perfectionné. Des trous de forage de 200m de longueur seraient préférables pour un meilleur contrôle des trous directionnels et un meilleur espacement.

Le procédé de fracturation hydraulique du charbon en tant que roches-réservoirs a été appliqué à plusieurs endroits afin d'augmenter l'échappement du gaz par le trou de forage, mais jusqu'à maintenant, ce procédé a été fait à partir de la surface seulement et avec un succès partiel.

Les mines du chantier de charbon de Sydney qui se prolongent sous la mer travaillent présentement dans deux couches gazeuses. Les tailles rabattantes modernes peuvent produire de larges tonnages mais le contrôle du méthane sur le front de taille se fait par ventilation (dilution) seulement. Ce système peut avoir comme effet de réduire la production de la taille à mesure que la mine avance à de plus grandes profondeurs. N'importe quelle quantité de méthane retirée de la couche avant l'extraction à longue taille aidera à atteindre une plus grande production.

Pour les chantiers de charbon de Sydney, le présent rapport recommande un programme de recherche en phases successives afin d'évaluer la faisabilité du pré-drainage du méthane. La phase initiale comporterait le forage, les tests de charbon dans les carottes de sondage et les tests in-situ pour l'évaluation des roches-réservoirs. La deuxième phase comporterait d'autres forages, des analyses d'échantillons et la surveillance des roches-réservoirs. Ces mêmes trous seraient utilisés durant la troisième phase quand les roches-réservoirs seraient stimulées par la fracturation hydraulique à partir de travaux souterrains.

Ce rapport recommande qu'un essai initial soit tenté dans "Phalen Colliery" parce que le développement de la taille rabattante y permet l'accès des chambres à longues tailles avant l'extraction; aussi le travail actuel se fait relativement près des entrées de tunnel et de plus, il se peut qu'il n'y ait pas de contraintes dans la couche Phalen à cause d'une trop grande exploitation dans la couche Harbour.

AR-124

PRE-DRAINAGE IN COAL MINES

SECTION I - INTRODUCTION

TABLE OF CONTENTS

1.1	GENERAL	1-1
1.2	CANADIAN REQUIREMENTS	1-1
1.3	TERMS OF REFERENCE	1-2
1.4	STUDY APPROACH	1-2
1.5	PROJECT TEAM	1-3
1.6	STANDARD UNITS	1-3

PRE-DRAINAGE IN COAL MINES

I.0 INTRODUCTION

I.1 GENERAL

The drainage of methane gas from coal seams and coal measure strata has been developed both as a safety measure to reduce the risk of explosion and as a source of energy. Within a coal mine, sources of emission may be from the coal seam itself, from roof and roof strata, from the floor and floor strata, from the worked and unworked areas and from the coal during cutting and transport. Depending on circumstances, the gas may be drained from the workings during and after mining, or from the areas where the coal has not yet been worked. Drainage from areas to be worked is termed pre-drainage. Pre-drainage may be by advanced excavation, drillholes within the seam, drillholes in the strata above and below the seam and drillholes from the surface or other seams. Suction may be applied to the holes to increase the efficiency of drainage and the holes may be stimulated by forms of hydrofracturing. Increases in the rate of coal production will cause an increase in the rate of gas released. In some cases, it is becoming essential to remove some of the gas prior to mining to reduce the risk of explosion and to keep ventilation velocities within manageable limits.

This study addresses the pre-drainage of gas from the coal seam itself.

I.2 CANADIAN REQUIREMENT

The off-shore mines on Cape Breton Island, operating at ever increasing distance from the shore and depth below the seabed, have quite distinct characteristics and operating constraints. Methane, from the coal ahead of the coal face in the seams being worked, flows through the seam into the coal face. More gas is released when the coal is cut, broken and transported.

Due to the location of the mine openings with respect to the production faces, there is a limit to which the air required to dilute the gas can be increased. As the depth and the rates of production increase, controlling methane on the faces by dilution alone could become impractical. Pre-drainage of the seam being worked may become the answer to this problem.

1.3 TERMS OF REFERENCE

The terms of reference for this report were:

- Conduct an in-depth review of reported techniques of pre-drainage of methane from coal seams in general and low permeability seams in particular; including a review of relevant drilling technologies and directional drilling controls together with gas flow stimulation techniques like hydrofracturing and foam stimulation. Summarize advantages and disadvantages of each method.
- Review the applicability of methods to Canadian coal mines in general and to Cape Breton collieries in particular.
- Select the most promising method for Cape Breton collieries and prepare a detailed program of testing in one of the collieries.

1.4 STUDY APPROACH

The in-depth literature review revealed very early in this study that gas pre-drainage had been carried out for a number of quite different reasons:

- produce saleable pipeline gas.
- reduce methane make during development.
- reduce outburst potential.
- reduce methane make during longwall extraction.
- solve particular, local ventilation problems.

The equipment used for drilling holes, draining water and/or gas varies greatly. Operational problems in a mine are solved in an ad hoc manner using available equipment. Where methane is the saleable product, it becomes quite obvious that capital funds were available for drilling and hole completion.

Section 3 gives the case histories of 19 gas pre-drainage examples. Section 4 deals with drilling of holes from underground coal mines and Section 5 covers hydrofracturing of coal. Section 6 is a discussion of the information in the preceding three sections. Section 7 addresses the test program at CBDC's Phalen and Lingan Collieries.

All of these sections are preceded by some general concepts on predrainage in Section 2.

1.5 PROJECT TEAM

This study was carried out under the direction of H.U. Bielenstein, Project Leader, by P.L. Wright of AMCL, Jeff Latos and Hal Anderson of NOWSCO Well Service Ltd.

1.6 STANDARD UNITS

	<u>UNIT</u>	<u>SYMBOL</u>
Gas Flows	litres per second	L/s
	Std. cubic metre per day	SCM/day
Lengths	metres	m
Diameters	millimetres	mm
Volumes	cubic metres	m ³
Pressures	kilopascals	kPa
Gas Content	cubic metres/tonne	m ³ /t
Permeability	millidarcy	md
Viscosity	millipascal seconds	mPa.s

PRE-DRAINAGE IN COAL MINES

SECTION 2 - PRE-DRAINAGE CONCEPTS

TABLE OF CONTENTS

2.1	INTRODUCTION	2-1
2.2	GAS CONTENT IN COAL	2-2
.1	State of Gas in Coal	
.2	Determination of Gas Content of Coal Seams	2-2
2.3	FACTORS CONTROLLING GAS RELEASE FROM THE WORKING SEAM	2-4
2.4	PROPERTIES WHICH EFFECT GAS PRODUCTION FROM THE COAL SEAM	2-5
.1	Sorption	2-5
.2	Permeability	2-5
.3	Fluid Pressure and Stress	2-6

TABLE

2-1	Methods of Estimation of Gas Content of Coal Seams
-----	--

LIST OF FIGURES

2-1	Methane Adsorption Isotherm for Coal
2-2	Relative Permeability Curves for Coal

PRE-DRAINAGE IN COAL MINES

2.0 PRE-DRAINAGE CONCEPTS

2.1 INTRODUCTION

Methane drainage has been practiced for many years in Europe, the U.S. and Australia. The present potential for high production rates in underground coal mines together with commercial use of methane gas has shown the need for pre-drainage prior to coal extraction. It is important to identify the need for drainage and the zone from which methane can be released before selecting a method of pre-drainage. Pre-drainage methods include drilling holes from either the surface or underground or by underground working. Possible sources of the methane may be from the coal seam from the strata above the seam and from the strata below the seam. Reservoirs of methane may exist in the above zones and also in the goaf or fractured rock mass after mining. The mining of more than one seam may cause gas to be liberated by destressing strata adjacent to the seam worked or being worked. Drainage may be effected by cross-measure drillholes either from surface or from underground or directionally controlled holes from within the seam itself.

This project addresses the requirements for drainage within the seam from underground workings. Before a drainage scheme is implemented, it is advisable to know how much gas is in the seam and whether the gas is in the coal substance or in the joints and cleats within the seam. The quantity of gas within the coal substance can be estimated by laboratory tests on core samples whereas the free gas in fractures must be determined by in situ tests. When the type and composition of gas is known e.g., CO₂, CH₄ and water, fluid movement in the coal substance can be tested in the laboratory and compared with fluid movement within the seam. The presence of water in the seam adversely affects the rate of gas release.

The method of mining and mine design can cause variation in stress around mine workings resulting in confinement of gas by high stress barriers both in immediate rib sides and above and below the seam and working.

2.2 GAS CONTENT IN COAL

.1 State of Gas in Coal

Coal and methane gas have a strong physical attraction. Methane is in a sorbed state rather than in a free gas state in the coal. In the context of in-seam drainage, the sorbed state is taken to include both adsorbed gas and absorbed gas. Adsorbed gas or water is the adhesion of a thin layer of molecules of gas or water to the surface of a substance. Absorbed gas or water is fluid taken into the pores of the substance.

Desorption is the release of gas or water from a substance, being the reverse of both adsorption and absorption. When water pressure equals the pressure at which the gas is sorbed, into coal solids or into solution, it is termed equivalent sorption.

The physical state of gas within the coal mass (seam) is, therefore, a combination of gas in the coal substance itself and in the fractures within the seam.

.2 Determination Of Gas Content of Coal Seams

The gas content of coal seams can be determined by direct or indirect methods.

Direct methods require in-situ material, usually intact core. The sample is sealed in a container; the gas that evolves from the sample

can be abstracted and measured. Crushing of the sample in the container may be required to release the gas within a reasonable time period.

Indirect methods are based on either empirical data obtained from existing mines which relate to measurements of gas from coal or from gas adsorption characteristics of coal under controlled pressure and temperature conditions.

Modern methods are reviewed by Lama and Bartosiewicz (1982), reproduced in this report as Appendix A. A summary of the methods described are shown in Table 2-1. They emphasize that the total gas content is difficult to assess accurately and is likely to be subject to error in the range of 20 to 50%. An investigator should be aware of possible inaccuracies especially as estimates are only as good as the source data. Results will be enhanced by including the following:

- Local conditions, e.g., fractures in coal, direction of cleat and efficiency of sealing hole.
- Coal sampling efficiency, e.g., length of core, recovery, time of cutting, withdrawing and efficiency of sealing samples.
- Water content of strata and core at time of sampling.
- Relative pressures and hydrostatic head.
- Relative temperatures.
- Boundary conditions.
- Composition of gas released.

In the case of a new seam or mining area, it is prudent to use both direct and indirect methods on samples from different locations within a drillhole and also from different horizons within a coal seam. This distribution can illustrate the range of gas contents possible within a seam.

2.3 FACTORS CONTROLLING GAS RELEASE FROM THE WORKING SEAM

The following factors influence the release of gas from the working seam:

- Gas content - the amount of gas in the coal.
- Composition of the gas - at a given partial pressure and temperature carbon dioxide is much more readily adsorbed than methane; methane itself is more readily adsorbed than nitrogen.
- Permeability effects - in addition to the permeability of the coal the rate of gas release is affected by the presence of water, the effect of abutment pressure differentials.
- Zones of influence - in addition to abutment zones the destressing of strata by working the seam or other adjacent seams either above or below will influence the rate of gas release. Zones of influence are affected by stress variations within the seam and surrounding strata. Boundary conditions may be induced by high stress abutments (Fisekci and Barron, 1975), but may be extended due to destressing by coal extraction, caving or workings in other seams (Boxho et al, 1980).
- Method and rate of mining - the mining method relating to the rate of breaking coal, plus the type of support and caving combined with the rate of extraction and advance will all affect the rate of gas release. The method and rate of mining affect the free faces exposed to desorb gas and also the time that gas can be desorbed in the mine. The extent to which the coal is broken and the size of the broken coal also has a significant effect on the gas flow.

In the process of determining the gas content of coal if the indirect method incorporating adsorption techniques is used, desorption rates of gas at specified temperature and pressure can be quantified. The rate of gas release cannot exceed the rate of desorption, but the desorption of gas from coal is proportional to the gas content, providing all other factors remain constant. (Hayes, 1982, Appendix B).

2.4 PROPERTIES WHICH EFFECT GAS PRODUCTION FROM THE COAL SEAM

.1 Sorption

Coal and methane gas have a strong physical attraction and the methane is therefore, in a sorbed state rather than in a free gas state. To free the sorbed gas from coal very low pressures must be achieved. According to the situation presented by Jones et al, (1986) if the reservoir pressure is reduced by 90% to 1380 kPa (200 psi) only 50% of the gas is released (Figure 2-1). For the same pressure reduction in a sandstone, where the gas is in a free state, 90% would be released.

.2 Permeability

Permeability is the ease with which fluid can flow through porous medium. The standard unit of measurement is the Darcy. This is based on a water rate of 1 cm/s under a pressure differential of one atmosphere per cm² at 20°C.

Large contrasts in permeability can exist for coal. The permeability along the cleats is much higher than through the matrix. Most of the flow therefore occurs along the cleats. The magnitude of this permeability will depend on both the cleat spacing and the cleat width.

The relative permeability is a reduction in the actual permeability of a single fluid phase resulting from the addition of another fluid phase. As shown in Figure 2-2, for coal, the relative permeability of water at 97% water saturation is down to 0.1 and at 90% water saturation is almost zero. That is, if the permeability of water in coal at 100% water saturation is 10 md then at 97% water and 3% gas, the actual permeability of water is 1 md. Figure 2-2 also shows that if the water is at its irreducible saturation (80% saturation) the relative permeability of the gas is 0.1 or 10% of its maximum at 0% water saturation.

These values are for flow through the rock matrix, although the same principal will apply to flow along the cleats. The relative permeability curves would change with cleat width.

One reason why the relative permeability curves are so steep is that the pore space in coal is composed of microspheres with radii less than $0.02 \mu\text{m}$. Generally, sandstone pore sizes range from 0.02 to $0.2 \mu\text{m}$. The small pore size increases the capillary pressure which effects the irreducible water saturation (generally 70% to 90%) and steepens the relative permeability curves.

.3 Fluid Pressure and Stress

The major fluid conductivity occurs along the cleat. There is an inter-relationship between fluid pressure, stress and drainage since opening and closing of the cleats will change the cleat permeability.

According to Gray (1987), cleat permeability is controlled by the effective stress. The effective stress is the total stress less the fluid pressure. Gray states that as the effective stress is reduced, the cleat opens and the coal permeability increases.

As fluid is produced from the coal, the fluid pressure drops, the effective stress increases and the cleat closes; permeability therefore drops. As the pressure drops, on the other hand, gas is desorbed and the coal shrinks; this reduces the total and the effective stress and increases permeability. A situation in Japan, however, has been found where very little shrinkage occurs with desorption. The coals became self sealing with production and pressure drop. Depending on the coal, stress changes associated with gas production can either increase or decrease permeability.

PRE-DRAINAGE IN COAL MINES

METHODS OF ESTIMATION OF GAS CONTENT OF COAL SEAMS
TABLE 2-1

Method	Requirements	Methods of Interpretation	Accuracy	Country	Method of Measurement	Applicability
DIRECT	Cores.	Requires knowledge of gas emission laws.	Depending upon time lost, core quality and applicability of gas emission laws. $\pm 50\% - \pm 20\%$	U.K. U.S.A. Australia	Volumetric gas emission at atmospheric pressure followed by crushing.	Good from surface boreholes and doubtful from underground boreholes, fractured cores etc. Degree of fracturing influences results.
DIRECT	Sampling of fractions.	Empirical.	Not Assessed.	Poland	Volumetric gas emission.	Existing mines.
INDIRECT Statistical	Lumps of coal.	Statistical analysis of gas content of lumps.	$\pm 20\%$, statistically measured values from face samples are consistently higher.	U.K.	Sample (30-40 mm size), collection at the face and estimation of gas contents.	For seams under mining, face sampling, applicability to highly variable and high rank coals not proved.
INDIRECT Adsorption Isotherms	Pressure measurement and sampling.	Direct reading for adsorption isotherms.	Depending upon pressure measurements and sampling. $- 10-20\%$.	Poland USSR Australia Germany	Volumetric techniques and gravimetric techniques.	For seams under mining and seams lying above or below.
INDIRECT Chemical Analysis	Proximate analysis of coal.	Empirical.	$- 30-100\%$.	USSR U.S.A. Poland	Chemical analysis.	New and existing mines.
INDIRECT	Exhaust ventilation sampling.	Subtraction techniques.	$\pm 20\%$.	Australia	CH ₄ or CO ₂ gas analysis (%).	Existing mines.
INDIRECT	Sampling of exhaust mine air.	Empirical.	$\pm 50\%$.	Germany U.S.A.	Volumetric gas emission.	Existing mines - general make of gas.

From: Lama and Bartosiewicz, 1982

AR-124

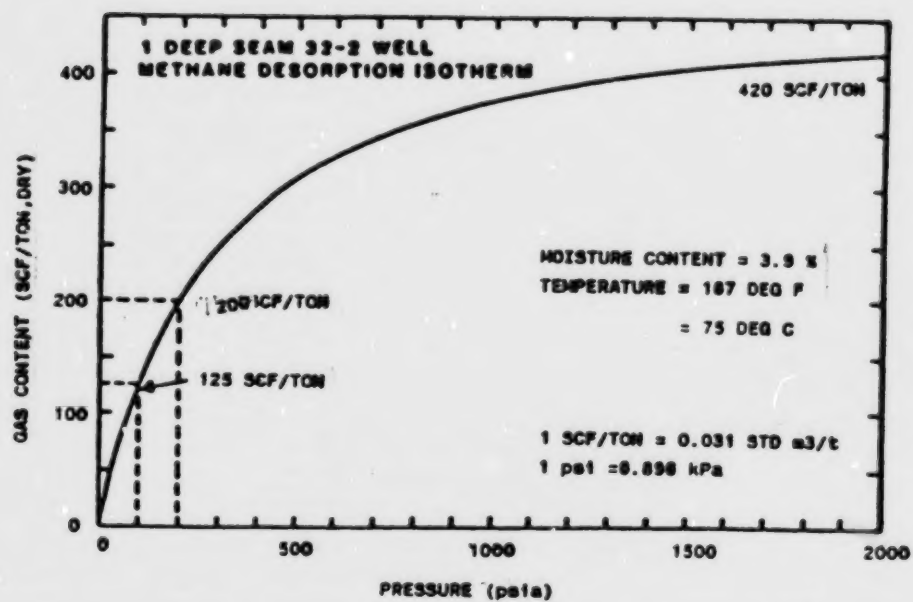


Figure 2-1
Methane Adsorption Isotherm for Coal
After: Jones et al, 1986

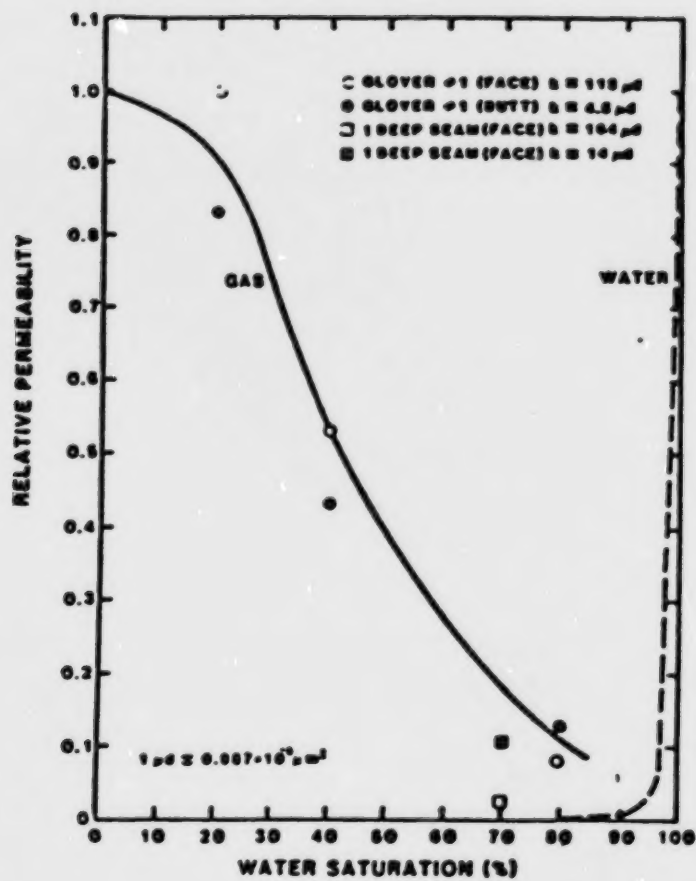


Figure 2-2
Relative Permeability Curves for Coal
From: Jones et al, 1986

PRE-DRAINAGE IN COAL MINES
SECTION 3 - CURRENT APPLICATIONS

TABLE OF CONTENTS

3.1	INTRODUCTION	3-1
3.2	AUSTRALIA	3-2
.1	Appin Colliery	3-2
.2	Westcliff Colliery	3-10
.3	Cook Colliery	3-15
.4	Leichhardt Colliery	3-17
.5	Metropolitan Colliery	3-23
.6	Hebburn Colliery	3-29
.7	Moura No. 2 Colliery	3-30
.8	Balmain Colliery	3-32
.9	John Darling Colliery	3-33
.10	MacQuarie Colliery	3-35
3.3	UNITED KINGDOM	3-36
.1	Cronton Colliery	3-36
.2	Golborne Colliery	3-40
.3	Silverdale Colliery	3-43
.4	Yorkshire Main	3-46
.5	Point of Ayr Colliery	3-47
3.4	UNITED STATES	3-50
.1	Sunnyside No. 1 Mine	3-50
.2	V.P. No. 5 Mine	3-54
.3	Soldier Canyon Mine	3-63
3.5	CANADA	3-65
.1	No. 26 Colliery	3-65

PRE-DRAINAGE OF COAL MINES SECTION 3 - CURRENT APPLICATIONS

LIST OF TABLES

- 3-A V.P. No. 5 Mine - Drainage Data
- 3-B No. 26 Colliery - Summary of Predrainage Holes

LIST OF FIGURES

- 3-1 Proposed Pattern of Advance Drainage Holes for Panel Development 3-7
- 3-2 Incremental Gas Flows into Holes Traversing Longwall Blocks 3-8
- 3-3 Emission Tests Reading and Roof Heights in the 3-Headings of "L" Panel Appin Colliery 3-9
- 3-4 Relationship Between Hole Diameter and Flow Rate 3-26
- 3-5 Emission Values During "A" Heading Drivage Relative to Intersection of Drainage Holes 3-27
- 3-6 Variation of CO₂ Flow Rate with Applied Suction from Drainage Holes - Using Nash Hytor Exhauster Pump. I NW Panel, Metropolitan Colliery 3-28
- 3-7 Cronton Colliery 3-38
- 3-8 Methane Drainage Holes 3-39
- 3-9 Golborne Colliery 3-42
- 3-10 Silverdale Colliery 3-45
- 3-11 Drainage Scheme - Point of Ayr Colliery 3-49
- 3-12 Sunnyside No. 1 Mine Hole Locations - 18 Dips 3-53
- 3-13 Location of Methane Drainage Holes 3-58
- 3-14 Trajectory of Oxy No. 5 Hole 3-59
- 3-15 Mine Plan and Location of Intercepted Holes 3-60
- 3-16 Desorption Curves for Virgin and Degassed Coal Samples 3-61
- 3-17 Effect of Inseam Drainage on Gas Desorption Rate 3-62

PRE-DRAINAGE IN COAL MINES

3.0 CURRENT APPLICATIONS

3.1 INTRODUCTION

Examples of previous experience in the pre-drainage of methane from coal seams in Australia, United Kingdom and North America are presented in this section in case history format. Gas flow stimulation of the coal seam has been researched and practised in the U.S. only. Drilling is covered in Section 4 and flow stimulation by hydrofracturing is discussed in Section 5.

The case histories include data on methods of determining gas content of both coal substance and coal mass (seams), methods of abstracting methane and equipment for separating water and methane and for safely transporting the gas.

3.2 AUSTRALIA

3.2.1 APPIN COLLIERY - NEW SOUTH WALES

Seam:	BULLI
- Depth:	550 m
- Thickness:	2.5 to 3.0 m
Purpose:	Pre-drainage for longwall panels
Type of Drilling:	In seam rotary
Hole Sizes:	
- Diameter:	43 mm
- Length:	45 m to 50 m
Drainage:	6 to 58 L/s
Hydrofracturing:	Hydrofracturing with sand proppant from surface holes
Effect of Hydrofracturing:	Not significant
Sources:	- Hargraves (1982b) - Battino and Hargraves (1982) - Lingard, Phillips and Doig (1982) - Stewart and Barro (1982)

.1 Relevance to Cape Breton Collieries

The main design features were that 43 mm diameter holes were drilled in coal at right angles to the rib 50 m long on 25 m spacing, releasing 15 L/s of methane per hole drilled. Advance holes 50 m long yielded 6 L/s. Flows fell off at approximately 1% per day. The direction of right angles to the rib rather than at right angles to the cleat (45° to rib) generated more gas per hole.

The abutment in the area was estimated to be 5 m in advance of developments and 15 m ahead of longwalls although other work has indicated the front abutment on longwalls to be 100 m in advance. Hydrofracturing was carried out from the surface. The hydrofracturing with sand proppant had no significant effect on roof conditions or on gas flows.

.2 Background

Several instantaneous outbursts of coal and gas have occurred at Appin Colliery with rising concentrations of methane at development faces, in bleeder headings and tailgate corners of longwall faces. Early trials using pre-drainage and post-drainage from above the goaf and into the floor were unpromising (Ryan and Beath, 1970). Workings are now at a depth of 550 m. The CH_4 content of the coal was $15 \text{ m}^3/\text{t}$ insitu and $1 \text{ m}^3/\text{t}$ in the run-of-mine coal at the surface (Hargraves, 1982b).

A surface hole drilled over the No. 2 longwall block had virtually zero flow until the face passed the hole position when natural flow reached 90 L/s and remained at the level of virtually pure CH_4 for several months.

Volcanic dykes and sills are widespread in the Upper Permian Bulli Seam.

.3 Permeability

The permeability variation of Appin coal was reported by Lingard et al. (1982):

	Permeability (md)		
	All Data	Perpendicular to Bedding Plane	Parallel to Bedding Plane
Mean	15.159	9.544	19.122
Standard Deviation	28.257	15.634	33.793
Standard Error of Means	5.321	4.714	8.448
Number of Cores	29	12	17
Co-efficients of Variation %	186	239	177

Effects of fissures on permeability:

	Permeability (md)	
	Visible Fissures	No Visible Fissures
Mean (all data)	26.554	1.135
Standard Deviation	33.869	0.507
Standard Error of Means	8.745	0.146
Number of Cores	16	13
Coefficients of Variation %	128	45

Visible fissures are those with a width greater than 0.05 mm. The confining pressure of the tests is 500 kPa. The low confining pressure permits high permeability. The wide range in values shows inhomogeneity of coal both in section and area. Stress would reduce the permeability of insitu coal from 10 to 100 times. There is also some uncertainty in the fracture/stress history of lump samples from the coal face which were used to obtain the cores.

Macropermeability of Appin Coal (Bulli Seam) averages 180 md for CH₄ and 100 md for CO₂.

Desorbable Gas from Appin Coal (Bulli Seam) at depth of 550 m is 13.8 m³/t.

Dynamic Elastic Modulus and Poisson's Ratio:

	Dynamic Young's Modulus, E (MPa)			
	All Data	Perpendicular to Bedding	Parallel to Bedding	Poisson's Ratio
Mean	2920	3304	2605	0.334
Std. Deviation	2555	3515	1238	0.109
Std. Error of Mean	586	1243	392	0.025
Number of Cores	20	9	11	10
Coefficient of Variation %	88	106	48	33

Effect of Fissures on Dynamic Elastic Modulus:

	Dynamic Young's Modulus (MPa)	
	Visible Fissures	No Visible Fissures
Mean (all data)	2124	3260
Standard Deviation	1559	2810
Standard Error of Means	697	779
Number of Cores	6	14
Coefficients of Variation %	73	86

.4 Drainage Experiments

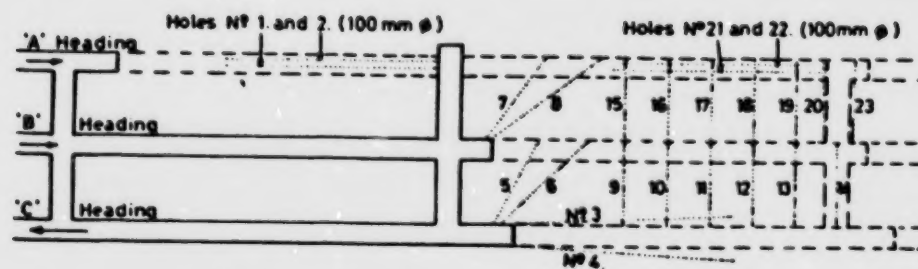
Patterns of 43 mm diameter holes up to 50 m in length and spaced 25 m apart were drilled into ribs yielded flows of 0.3 L/s/m. A group of 15 holes of average length 45 m discharged 58 L/s (0.09 L/s/m) through small individual hoses into the return. As inbye holes are drilled the flow falls off in the outbye holes they can be disconnected and be replaced by the inbye holes. The drainage holes are close enough to be inter-dependent.

At Appin the front abutments were estimated to be 5 to 15 m ahead of the face, but other work has shown abutments to be 100 m ahead of longwall faces. Holes of 43 mm diameter were drilled in the centre of pillars gave a maximum flow of 6.1 L/s. Generally flows fell off at about 1% per day. The proposed pattern of advance drainage holes for panel development are shown on Figure 3-1. The two 100 mm diameter holes are drilled to bleed away from cut-through faces into returns. These were introduced after ventilating pressure surveys established an appropriate pressure drop when an exhaust fan is in use in the advancing face. The holes appear to be successful.

Incremental flows into holes traversing longwall blocks at Appin Colliery are shown on Figure 3-2.

.5 Hydrofracturing Tests

An attempt at degasification of the Bulli Seam was made by hydrofracturing with sand proppant using a drill hole from the surface. The frac'd area was mined through by underground workings after completion of the drainage trials. To assess the success of the tests, the roof condition during development was monitored to observe any deterioration and gas emissions were monitored during development to identify changes in gassiness near the drainage hole. Results of the monitoring are shown on Figure 3-3 - indicating no significant change. Fluorescein dye was used in the frac and cuttings were subjected to ultra violet examination in the laboratory. The pillar area underground near the frac was also scanned using a portable ultra violet source. Both ultraviolet tests gave negative results - the induced fracture could not be located.



HOLES No.1,2,21,22 ARE 100 mm DIAM.
REMAINDER ARE 43 mm DIAM

Figure 3-1
Proposed Pattern of Advance Drainage Holes for Panel Development
Source: Battino and Hargraves (1982)

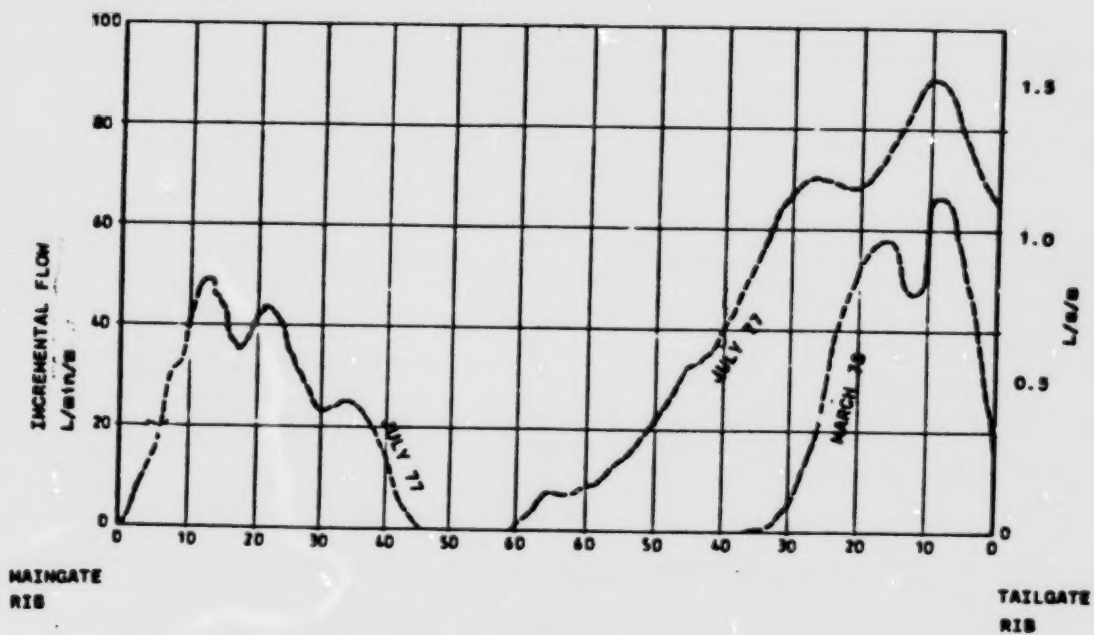


Figure 3-2
Incremental Gas Flows into Holes Traversing Longwall Blocks
Source: Battino and Hargraves (1982)

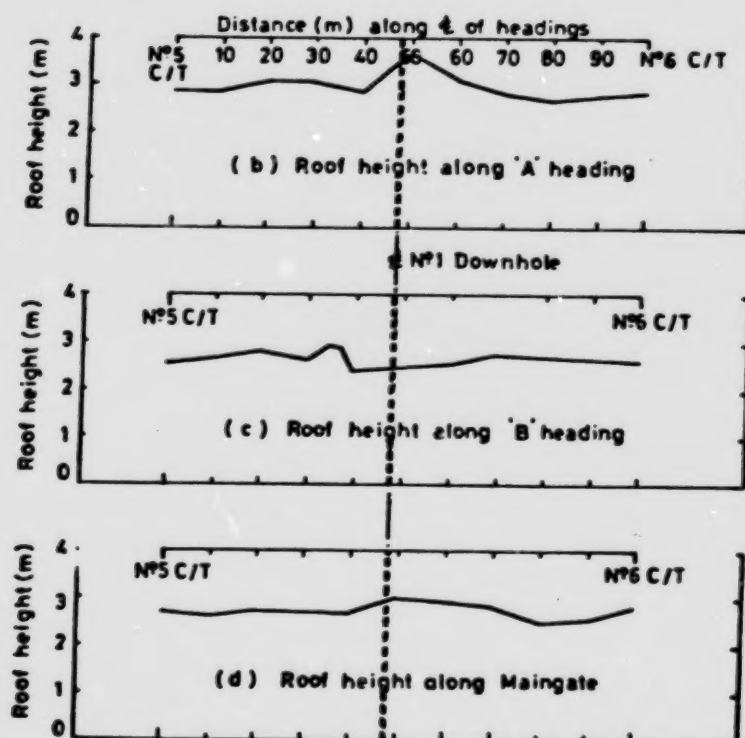
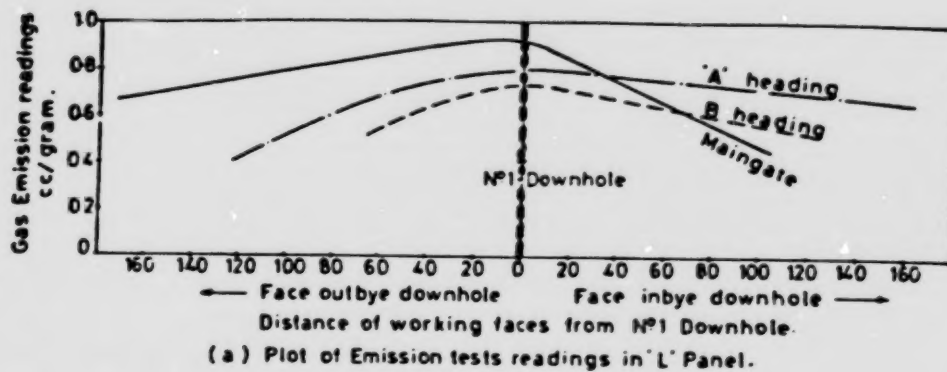


Figure 3-3
Emission Tests Reading and Roof Heights in
the 3-Headings of "L" Panel Appin Colliery
Source: Battino and Hargraves (1982)

3.2.2 WESTCLIFF COLLIERY - NEW SOUTH WALES

Seam:	BULLI
- Depth:	450 m
- Thickness:	2.3 to 3.0 m
Purpose:	Pre-drainage and methane drainage plus outburst prevention
Type of Drilling:	Rotary. Atlas Copco Diamec 250 Rotary and Gardner Denver Acker Big John
Hole Sizes:	
- Diameter:	50 mm reamed to 85 mm
- Length:	Up to 471 m
Drainage:	Methane by suction into 250 mm and 350 mm ranges
Hydrofracturing:	None
Sources:	Hayes (1982) Lingard et al (1982) Marshall et al (1982)

.1 Relevance to Cape Breton Collieries

The research and development at West Cliff Colliery are guidelines for a pre-drainage study in Cape Breton.

- Determination of gas content
- Determination of permeability
- Determination of zones of influence
- Specification of drill holes and geometry
- Identification of preferred drill
- Specification of drainage installation
- Problems encountered and remedial measures taken

.2 Background

In 1980, a full scale gas drainage system was commissioned at West Cliff Colliery to drain gas from the Bulli Seam.

The trials of drilling holes and testing the composition of coal together with identifying stress concentrations (abutment zones) have shown that methane drainage is successful in controlling gas emission rates at the face in a seam under extraction by pre-draining the seam gas using in-seam horizontal holes. The drainage also had a major effect on reducing the intensity of outbursts at shear zones.

.3 Permeability

The permeability of West Cliff coal was reported by Lingard et al. (1982):

	<u>Permeability (md)</u>		
	<u>All Data</u>	<u>Perpendicular to Bedding Plane</u>	<u>Parallel to Bedding Plane</u>
Mean	3.193	4.864	0.965
Standard Deviation	5.831	7.202	1.225
Standard Error of Means	1.122	1.859	0.369
Number of Cores	28	16	12
Coefficients of Variation %	183	148	127

Effects of fissures on Permeability:

	<u>Permeability (md)</u>	
	<u>Visible Fissures</u>	<u>No Visible Fissures</u>
Mean (all data)	10.702	0.690
Standard Deviation	7.626	0.946
Standard Error of Means	3.113	0.212
Number of Cores	7	21
Coefficients of Variation %	71	137

Visible fissures are those with a width greater than 0.05 mm.

The confining pressure of the tests is 500 kPa. The low confining pressure permits high permeability. The wide range in values shown inhomogeneity of coal both in section and area.

Dynamic Elastic Modulus and Poisson's Ratio:

	<u>Dynamic Young's Modulus, E(MPa)</u>			<u>Poisson's Ratio</u>
	<u>All Data</u>	<u>Perpendicular to Bedding</u>	<u>Parallel to Bedding</u>	
Mean	3636	3779	3399	0.260
Standard Deviation	1604	1752	1286	0.135
Standard Error of Mean	288	402	389	0.024
Number of Cores	32	20	12	32
Coefficient of Variation %	44	46	38	52

Effect of Fissures on Dynamic Elastic Modulus:

	<u>Dynamic Young's Modulus, E(MPa)</u>	
	<u>Visible Fissures</u>	<u>No Visible Fissures</u>
Mean (all data)	3252	3744
Standard Deviation	1587	1592
Standard Error of Means	648	325
Number of Cores	7	25
Coefficients of Variation %	49	43

.4 Drainage Experiments

Pre-Drainage Parameters Bulli Seam, West Cliff Colliery

<u>Parameter</u>	<u>Value</u>	
Maximum Gas Pressure	2670	kPa
Gas Content	13 m ³ /t	(Raw Coal)
Zone of Fracturing Around Roadways	6 - 8 m	
Sphere of Influence of Holes in Coal	27 - 60 m	
Initial Flow Rate (Pure methane at free flow)	0.04 L/s/m	
Flow Rate at a Suction of 20 kPa (150 mm of Hg)	0.08 L/s/m	

Source: Marshall et al (1982)

The first phase of investigation included determination of gas content, in-seam gas pressure, optimum suction, area of influence of holes placed in coal for pre-drainage. Local geological factors affecting flow rates were considered along with stress and rock properties, mining method, and the development of fractures and relaxation zones around mine excavations.

The drilling phases included variations in geometry of drill holes and utilization of different drilling machines. The following drilling machines were used and achieved maximum hole length:

- Acker Big John - 471 m
- Atlas Copco Diamec 250 - 174 m
- Edeco - 160 m
- Gardner Denver - 40 m

The Big John trial also incorporated the use of stabilizers and monitoring of both azimuth and inclination using the Eastman single shot survey tool. Experience in the Big John holes in maintaining both horizon and azimuth was not encouraging. Major problems occurred

when mylonitic zones were encountered causing the stabilization of the drill string to be ineffective due to blockage around the stabilizers. This slowed down the penetration and stalled the machine. The drill could operate through a stuffing box. The most economic bits were tungsten carbide 83 mm diameter 4 wing drag bit, 83 mm assembly drag bit and 80 mm pilot with reamer bits (Hebblewhite et al, 1982).

The preferred system is to use the Atlas Copco Diamec 250 machine proven to drill 174 m in coal and 120 m in rock. The geometry of the holes determined on flow rates and lead time was:

Optimum spacing	- 12 m to 18 m
Length	- 70 m to 90 m
Hole Diameter	- 50 mm
Hole Collar Reamed to	- 85 mm for depth of 9 m
Suction	- 25 to 30 kPa
Size of Drainage Pipe	- 250 mm diameter

Life of holes approximately 4 months.

The 9 m standpipes forming the collar were PVC carbon impregnated anti-static pipe.

In addition to the in-seam drainage, floor holes were drilled at 57 mm diameter with the top 6 m reamed to 100 mm. The average length of floor holes was 100 m with 6 being operative at any one time.

3.2.3 COOK COLLIERY, NEW SOUTH WALES

Seam:	Castor
- Depth:	250 m
- Thickness:	3.0 m
Purpose:	Prevention of outbursts
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	43 mm
- Length:	21 m to 25 m
Drainage:	2.9 to 1.25 L/s
Hydrofracturing:	None
Sources:	Battino and Hargraves (1982)

.1 Relevance to Cape Breton Collieries

Flow rates from 43 mm diameter holes 20 m in length drilled perpendicular to cleats were marginally higher than those from holes drilled parallel to cleat. Maximum gas pressure (900 kPa) was registered at a hole depth of 25 m.

.2 Background

Gas problems were experienced in the 3 m thick Castor seam at a depth of 250 m. During continuous mining small cones of coal burst off the face and face corners. In another area, a fault intersection resulted in emission of methane. The fault intersection was sealed off with a barricade and the gas was piped off. Experiments in pre-drainage from the seam were then carried out.

.3 Permeability

Macro permeability of samples from the Castor Seam averaged 190 md for CH₄ and 510 md for CO₂.

.4 Drainage Experiments

43 mm diameter rotary holes were drilled within the seam both at right angles to the cleat and parallel to the seam. The maximum pressure of 900 kPa was found at a hole depth of 25 m. Flow rates from holes 21 m in length ranged from 1.4 to 2.9 L/s perpendicular to the cleat and 1.25 L/s parallel to the cleat.

Flow rates from the holes in the centre of the seam were higher than from holes near the top and the bottom of the seam.

All of the above results should be considered in a Cape Breton pre-drainage program.

3.2.4 LEICHHARDT COLLIERY, NEW SOUTH WALES

Seams:	Gemini	Aries
- Depth:	380-430 m	340-390 m
- Thickness:	6 m	3 m
Purpose:	Outburst control and degassing of seam	
Type of Drilling:	Rotary	
Hole Sizes:		
- Diameter:	43 mm	
- Length:	6 m to 60 m (Generally under 30)	
Drainage:	0.17 to 0.74 L/s	
Hydrofracturing:	Foam stimulation in surface wells	
Effect of Hydrofracturing:	None	
Sources:	Battino and Hargraves (1982) and Lingard et al (1982)	

.1 Relevance to Cape Breton Collieries

Wet conditions were addressed together with details of assessing gas content. Tests included drill holes up to 300 mm in diameter, but were generally 43 mm. Flow rates increased with diameter but not in direct proportion.

.2 Background

The mine had a gassy environment with CH_4 and there were instantaneous outbursts of coal and gas, some without faulting. Tests were conducted in the Permian age Gemini seam (6 m thick) which is 40 m below the Aries seam (3 m thick). The workings are 380 m deep. Major cleats contributed to natural drainage of the coal along the perimeter of workings (Hargraves, 1982b). The gas content of the coal was 95% methane and 5% CO_2 . The coal is coking coal of low ash content and rank of approximately 1.24% maximum vitrinite reflectance.

.3 Permeability

The permeability of Leichhardt coal was reported by Lingard et al. (1982):

	Permeability (md)		
	All Data	Perpendicular to Bedding Plane	Parallel to Bedding Plane
Mean	35.582	64.354	6.811
Standard Deviation	91.451	122.579	6.751
Standard Error of Means	25.364	50.043	2.756
Number of Cores	14	7	7
Coefficients of Variation %	257	191	99

Effects of fissures on permeability:

	Permeability (md)	
	Visible Fissures	No Visible Fissures
Mean (all data)	59.855	3.218
Standard Deviation	115.110	3.796
Standard Error of Means	43.508	1.697
Number of Cores	8	6
Coefficients of Variation %	192	118

Visible fissures are those with a width greater than 0.05 mm. The confining pressure of the tests is 500 kPa. The low confining pressure permits high permeability. The wide range in values shows inhomogeneity in the coal. There is always uncertainty in obtaining samples from lump coal at the face. Insitu stress could reduce the permeability by 10 to 100 times.

The macro permeability of samples from the Leichhardt Coal Gemini Seam averaged 8 md for CH₄ and 10 md for CO₂.

Dynamic Elastic Modulus and Poisson's Ratio:

	<u>Dynamic Young's Modulus, E(MPa)</u>			<u>Poisson's Ratio</u>
	<u>All Data</u>	<u>Perpendicular to Bedding</u>	<u>Parallel to Bedding</u>	
Mean	938	794	1065	0.335
Std. Deviation	914	482	1153	0.136
Std. Error of Mean	244	197	436	0.035
Number of Cores	15	7	7	15
Coefficient of Variation %	97	61	108	39

Effect of Fissures on Dynamic Elastic Modulus:

	<u>Dynamic Young's Modulus, E(MPa)</u>	
	<u>Visible Fissures</u>	<u>No Visible Fissures</u>
Mean (all data)	680	1233
Standard Deviation	496	1160
Standard Error of Means	187	474
Number of Cores	8	7
Coefficients of Variation %	73	94

.4 Determination of Original Gas Content (Gray, 1982)

The gas content is defined as gas in pore-space, in solution in seam water and in sorption on the coal. Maximum gas content is limited by the storage capacity of the coal and the gas available for storage.

The two methods recorded to measure gas content of coal were:

- The direct core desorption test measuring gas released in time from a known mass in a cylinder.

- The measurement of seam fluid pressure and its comparison with a sorption isotherm determined in the laboratory. To make an assessment as to whether the fluid pressure is water pressure or gas pressure, a pressure bleed-off of gas rather than just water should be monitored. The monitoring should continue as pressures can in most cases, be related to gas content.

.5 Concept of Test Design

- Gas Material Budget - the initial gas in place less the gas produced equals the gas left.
- The continual monitoring of gas insitu either by core removal and testing or to monitor pressure by pressure sensors or by borehole seals.

.6 Method Adopted at Leichhardt Colliery

The initial gas content was found by measuring the pressure of the gas in the borehole using hydraulic packers to seal the end of the hole at various depths during drilling.

To examine possible leakage near the face of a hole, and to examine where gas was being emitted, incremental flow tests were done along the hole. The tests were carried out by sealing the borehole at various depths with a packer on the end of a conduit. Flows and pressures are measured at the mouth of the conduit, generating flow/unit length relationships along the hole. The flow/unit length can be integrated with time to provide the cumulative quantity drained. The material budget can then be balanced: Gas left equals initial gas in place less gas produced.

.7 Drainage Experiments

Drainage experiments included both in-seam boreholes within the Gemini seam and surface wells stimulated by foam hydrofracturing in an area 430 m deep outside the colliery working area. The in-seam drainage was described by Battino and Hargraves (1982) and the well drainage by Stewart and Barro (1982). A complete study of gas drainage was reported by Gray (1982).

In the Gemini seam, four 43 mm holes from an arched roadway 4 m to 21 m in length yielded 0.23 L/s with a maximum pressure of 9 kPa. Six 43 mm holes drilled from a rectangular roadway ranging from 6 m to 30 m in length yielded flows from 0.17 to 0.73 L/s.

.8 Gas Pressure Studies

Within the drainage experiments at Leichhardt Colliery, a study was made of gas pressures by using triple packers and "grouted in" pressure sensors in 43 mm diameter holes. Three areas of the mine were tested (Gray, 1982). Pressure distribution was measured using incremental packers. The along cleat holes tended to destress a shadow zone of a lower pressure along the cleat than holes drilled at right angles to the cleat. Pressures were recorded up to 2500 kPa close to the face in outburst conditions.

Holes were drilled 43 mm in diameter and up to 60 m in length.

In the Aries seam, a 43 mm hole drilled in the seam for 7.5 m yielded 0.2 L/s, but it was noted that the site had been exposed for eight years. The maximum gas pressure recorded was 70 kPa.

.9 Results

- The tests showed that 43 mm diameter drainage holes are efficient.
- The gas content of the coal should be determined.
- Grouted in pressure sensors can be effective for observations over several years.
- Pressure and flows can change over time, possibly by interhole leakage between close-spaced holes.
- High pressures measured in drillholes across cleat compared with lower pressures in drillholes along the cleat tied in with the formation of outburst core in the high pressure cross-cleat side of a heading.

3.2.5 METROPOLITAN COLLIERY, NEW SOUTH WALES

Seam:	BULLI
- Depth:	450 m
- Thickness:	
Purpose:	Reduce CO ₂ outburst problem
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	43 mm to 300 mm + 1 at 600 mm
- Length:	34 m to 80 m
Drainage:	0.3 L/s open hole
Hydrofracturing:	None
Sources:	Hargraves (1982b) Battino and Hargraves (1982) Hayes (1982)

.1 Relevance to Cape Breton Collieries

- Holes 43 mm diameter at 10 m and 7 m centres were used to drain entry pillars.
- Shot firing with pulsed infusion in 43 mm holes replaced open holes of 300 mm diameter.
- Safe limit to mine re outburst of gas conditions at Metropolitan 1.2 m³/tonne (this is not related in literature to gas flow).

.2 Background

Metropolitan Colliery has had a long history of CO₂ gas outbursts. The Bulli seam is 450 m deep.

There have been seven fatalities from 1895 to 1954 and the high rank coal leads to difficulties in drainage. Battino and Hargraves (1982) observe that since the gas state can be better defined than the stress state, and can be monitored, the alleviation of the problem lies in partially removing the gas down to a benign level. Virgin coal has a gas content of 15 m³/t of CH₄ and 40 m³/t of CO₂.

Desorbable Gas (Battino and Hargraves, 1982) at depth of 450 m is 7.5 m³/tonne.

The macro permeability of the Bulli Seam averages 90 md for CH₄ and 100 md for CO₂.

.3 Drainage Experiments

Heavy drilling equipment was used in 1969 to drill holes 300 mm diameter 80 m in advance of the face. The method was superseded by drilling 43 mm diameter holes and pulsed infusion shot firing using submarine explosive under reticulated (about 1500 kPa) water infusion conditions for degassing.

Tests using different hole sizes were made and results indicated that flows increased with the diameter of hole but not in direct proportion (Figure 3-4).

A test was designed to check the effect of drillhole spacing. 43 mm holes were drilled on 10 m and 8 m spacing into an area of future development. A distinct difference of methane make was recorded during driveage of the third heading (Figure 3-5). Later holes up to 130 m in length were drilled on 7 m spacings.

In view of low flows from the holes, an exhausting experiment was carried out in which nett CO₂ flows are examined against leakage (Figure 3-6).

The paper discussed the need for the longer holes and also the development of 43 mm bit air motor driven airleg mounted drill with E rods and water flushing now capable to drill 130 m. One hole was reamed to 100 mm diameter with the same equipment. In multi-hole situations, infusion in a centre hole can be by compressed air in the case of CO₂ and N₂ or water in the case of CH₄ to accelerate drainage.

.4 Results

- Holes 43 mm in diameter were drilled up to 130 m using air-leg mounted drill using E rods and water flushing.
- In the low permeability hole spacing was reduced from 10 m to 7 m.
- Suction was necessary to assist in gas flow.
- In wet holes, flow increased initially and reduced to 20% of the initial flows in 40 to 170 days depending on the size of pillar (23 m to 66 m).

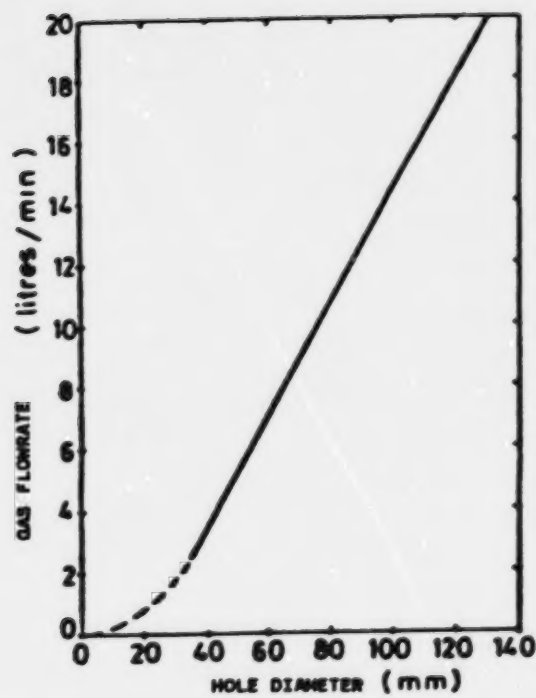


Figure 3-4
Relationship Between Hole Diameter and Flow Rate
After Battino and Hargraves (1982)

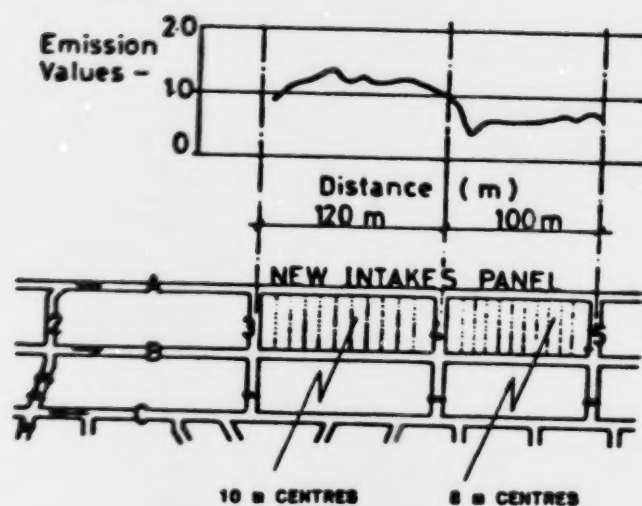


Figure 3-5
Emission Values During "A" Heading Drivage
Relative to Intersection of Drainage Holes
After Battino and Hargraves (1982)

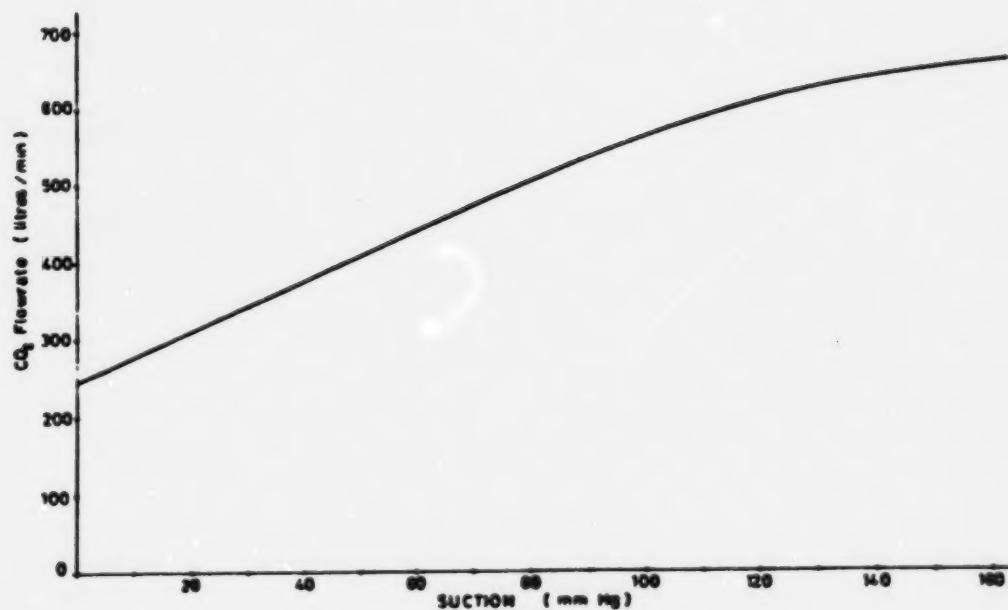


Figure 3-6
Variation of CO₂ Flow Rate with Applied
Suction from Drainage Holes - Using
Nash Hytor Exhauster Pump. I NW Panel, Metropolitan Colliery
After Battino and Hargraves (1982)

3.2.6 HEBBURN COLLIERY, NEW SOUTH WALES

Seam:	GRETA
- Depth:	
- Thickness:	5.5 m
Purpose:	Gas drainage CH ₄ and H ₂ S
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	43 mm
- Length:	15 m
Drainage:	3.3 L/s open hole pressure 1400 kPa 10.0 L/s with suction
Hydrofracturing:	None
Sources:	Hargraves (1982b)

.1 Relevance to Cape Breton Collieries

Workings were in the bottom 3 m of a 5.5 m split seam. A mudstone band approximately 1 m thick prevented drainage of top coal. Top coal was intersected by boreholes and gas drained.

3.2.7 MOURA NO. 2 COLLIERY, NEW SOUTH WALES

Seam:	Moura D
- Depth:	200 m
- Thickness:	
Purpose:	Prevention of outburst of CH ₄ during the course of continuous mining
Type of Drilling:	
Hole Sizes:	
- Diameter:	43 mm and 100 mm
- Length:	18 m and 21 m
Drainage:	Decrease in gas make during mining
Hydrofracturing:	None
Sources:	Battino and Hargraves (1982)

.1 Relevance to Cape Breton Collieries

Holes drilled 21 m long appeared to reduce the gas during mining.

.2 Background

The gas in the Moura "D" seam in the Bowen Basin, at a depth of 200 m is virtually pure CH₄. Gas burst cones appeared during continuous mining. The initial drainage tests were conducted as a preventative measure.

.3 Drainage Tests

Holes both 43 mm and 100 mm were drilled 18 m and 21 m into the face. The apparent effect of the drilling was to spread the total emission more regularly. The test results are shown in the following table.

Test No.	Hole Specification				CH ₄ Make Whilst Machining (L/s)
	43 mm Dia.	Length (m)	100 mm Dia.	Length (m)	
Test 1	0	-	0	-	86.3
Test 2	2	6	1	21	74.5
Test 3	1	21	-	-	-
	2	3	1	18	-
Test 4	1	18	2	21	65.2
	1	18	0	-	83.2

Relation Between Gas Make During Machining
and Size and Number of Holes in Heading Face
From: Battino and Hargraves (1982)

3.2.8 BALMAIN COLLIERY, NEW SOUTH WALES

Seam:	BULLI
- Depth:	880 m
- Thickness:	
Purpose:	Gas production
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	
- Length:	
Drainage:	Gas production from cross measure hole 1505 m long intersection 20 seams 463 L/s
Hydrofracturing:	None
Sources:	Hargraves (1982b)

.1 Relevance to Cape Breton Collieries

Gas production from abandoned coal mines requires stringent safety precautions to avoid explosive atmospheres.

.2 Background

Two 5.5 m brick lined shafts intersection the Bulli seam at 880 m in 1902. Production until 1931 was 400 t/day. A vertical hole was drilled 1505 m for natural gas production intersecting 20 coal seams over 1386 m. In 1945, an attempt was made to seal the shafts 13 m below the surface when an explosion destroyed the headframe and killed three men. In 1946 when the shafts were re-sealed, the flow of CH₄ was 35 L/s which declined to 23 L/s in 1950 when gas production ceased.

3.2.9 JOHN DARLING COLLIERY, NEW SOUTH WALES

Seam:	Borehole
- Depth:	-
- Thickness:	-
Purpose:	Gas drainage for longwall mining
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	43 mm
- Length:	70 to 75 m
Drainage:	Up to 5 L/s per hole
Hydrofracturing:	None
Sources:	Battino and Hargraves (1982)

.1 Relevance to Cape Breton Collieries

- Hole sizes 43 mm, 70 to 75 m long.
- Gas flows reduced significantly after 14 days.

.2 Background

John Darling Colliery is one of the deeper mines in the Newcastle area, and has had problems with increasing gas make of CH_4 in the development for a longwall. There are no coal seams below the Borehole seam, but overhead is the Dudley at 30 m and the Victoria Tunnel at 60 m, both unmined.

.3 Drainage Tests

Drainage tests were conducted in the Borehole seam in the Newcastle area from the development for the 75 m wide No. 1 Longwall. Holes 43 mm in diameter were drilled 70 to 75 m in advance of the development. Flow measurements are shown in the following table.

Position of Drainage Hole	Hole Length (m)	Gas Flow Rate (L/s)		
		24 Hrs. After Drilling	7 Days After Drilling	14 Days After Drilling
No. 30 C/T	73	1.5	0.6	0.5
No. 31 C/T	75	1.2	1.2	1.0
No. 32 C/T	50	1.2	0.2	0.1
No. 33 C/T	50	1.7	0.2	-
No. 34 C/T	70	5.0	2.3	1.7
No. 36 C/T	72	3.0	2.0	1.2
No. 40 C/T	70	0.7	0.5	0.2
No. 44 C/T	70	0.8	0.5	
No. 48 C/T	70	0.8	0.5	
No. 51 C/T	71	0.5	-	-

Gas Flow Measurements from
Drainage Holes in the No. 1 Longwall Block
Borehole Seam, John Darling Colliery
From: Battino and Hargraves (1982)

Increasing the spacing between holes did not increase gas flows. The Australian author's comment "the results do not give great promise of in-seam drainage in the Borehole Seam." It was recommended that post drainage should be examined.

.4 Macroporosity

Samples are trepanned from lump, both parallel and perpendicular to bedding, and are encapsulated in a length of steel pipe and epoxy resin. The ends of the coal are faced and the CH₄ and later CO₂ is passed through the coal to measure macroporosity.

The permeability of the Borehole Seam averages 9 md for CH₄ and 12 md for CO₂.

3.2.10 MACQUARIE COLLIERY, NEW SOUTH WALES

Seam:	Young Wallsend
- Depth:	-
- Thickness:	-
Purpose:	Pre-drainage for longwall workings
Type of Drilling:	Rotary
Hole Sizes:	
- Diameter:	43 mm
- Length:	44 to 117 m
Drainage:	Less than 0.8 L/s
Hydrofracturing:	None
Sources:	Battino and Hargraves (1982)

.1 Relevance to Cape Breton Collieries

- 43 mm holes in-seam and up to 117 m long.
- With seams above and below a post drainage test is planned.

.2 Background

As the deepest BHP colliery in the Newcastle area, ventilation problems have been experienced associated with faults and dykes. The Victoria Tunnel Seam, 2.5 m thick, lies 10 m above the Young Wallsend Seam and the Borehole Seam 2.3 m thick, lies 16 m below.

.3 Drainage Tests

As a pre-drainage test, two 43 mm diameter holes were drilled in the Young Wallsend Seam in the vicinity of the No. 1 longwall. One hole intersected a dyke at 44 m depth. The other hole, 117 m long, produced less than 0.8 L/s of gas.

3.3 UNITED KINGDOM

3.3.1 CRONTON COLLIERY, LANCASHIRE

Seam:	Wigan Four Feet
- Depth:	-
- Thickness:	1.8 m
Purpose:	Degassing of single entry development drainages 1200 m long for retreat mining
Type of Drilling:	50 mm Rotary
Hydrofracturing:	None
Sources:	Highton (1982a, 1982b)

.1 Relevance to Cape Breton Collieries

- 40% of the total methane (48 L/s of a total 120 L/s) was emitted in the 20 m nearest the face of the entry.
- Drilling pattern as shown in Figure 3-7 was developed.
- Two 50 mm diameter holes angled at 15° and 30° were driven on each side. Provided they were maintained 20 m in advance and 20 kPa suction applied, the polluting effect was reduced to 10 L/s of methane.
- Methane captured from the system varied between 10 and 15 L/s per hole with a total system drainage of between 40 and 60 L/s. Suction was maintained if purity was in excess of 20% methane. 15° holes had more effect on methane emitted from the face of the drivage, but 30° holes appeared to remain active longer and had some control of methane emitted out by the heading face.
- Additional 50 mm diameter holes drilled to a depth of 100 m at right angles to the heading at intervals of 50 m partially controlled outbye emission. By the time the entry had advanced 1000 m, the right angle holes were capturing in excess of 150 L/s.
- At the 1200 m drivage approximately 50% out of a total make of 370 L/s methane was captured by the in-seam drilling. The remaining 50% was effectively controlled by a twin 900 mm auxiliary ventilation system delivering fresh air at 15 m³/s at the inbye end of the drivage.

- Back return ventilating system for retreating longwall - Figure 3-8. The purpose of the sealed return was twofold: to maintain drainage after the face was worked and to create an extinctive atmosphere behind the seals to prevent spontaneous combustion. The system is generally used in the U.K. in conjunction with cross measure drainage holes.

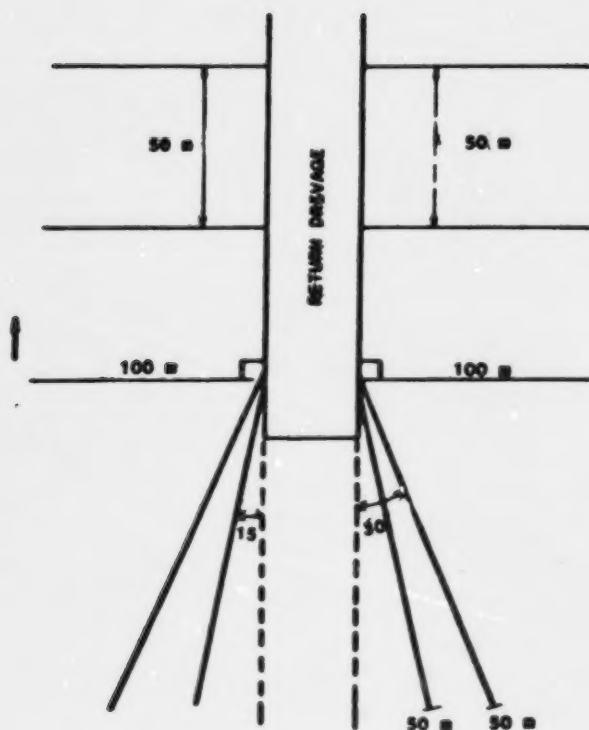


Figure 3-7
Cranton Colliery
Source: Highton (1982a)

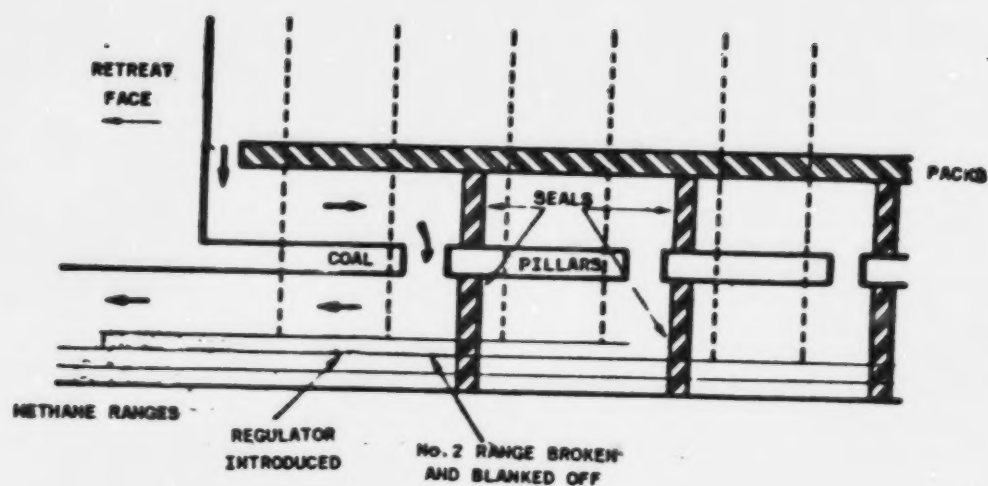


Figure 3-8
Methane Drainage Holes
Source: Highton (1982b)

3.3.2 GOLBORNE COLLIERY, LANCASHIRE

Seam:	Lower Florida, Section 2
- Depth:	400 m
- Thickness:	-
Purpose:	Degassing of advancing Longwall
Type of Drilling:	In-seam rotary in addition to cross measure over goaf
Hydrofracturing:	None
Sources:	Highton (1982a)

In 1979, there was an explosion of methane in development headings at Golbourne Colliery.

In the advancing longwall panel, the practice was to drill cross measure holes up to the Ince Six Feet seam, 40 m above and down holes to the Pigeon House Seam 30 m below. These holes were driven as close as 10 m apart. The system drained up to 600 L/s pure methane. Ventilation quantities of 15 to 20 m³/s were insufficient to dilute to less than 1.25% methane (Figure 3-9).

.1 Relevance to Cape Breton Collieries

- Tests showed that as the face advanced extra methane was being emitted from the seam itself and not from the roof and floor.
- A program of in-seam drilling was implemented for holes to be drilled on the high side of the face at 10 m intervals. Holes were 50 mm diameter and 60 m long. Standpipes were 10 m long steel pipe, 52 mm internal diameter grouted in with quick drying cement.
- Faulty grouting permitted one standpipe to be blown-out by gas pressure.
- Gas pressures built up to over 2400 kPa but gas flows only averaged 15 L/s from each hole.

- Tests showed that suction of 15 kPa was necessary for the 15 L/s flow.
- With a suction of 15 kPa on each hole, the sphere of influence from adjacent holes 15 m apart did not overlap, but at 10 m apart, interaction took place.

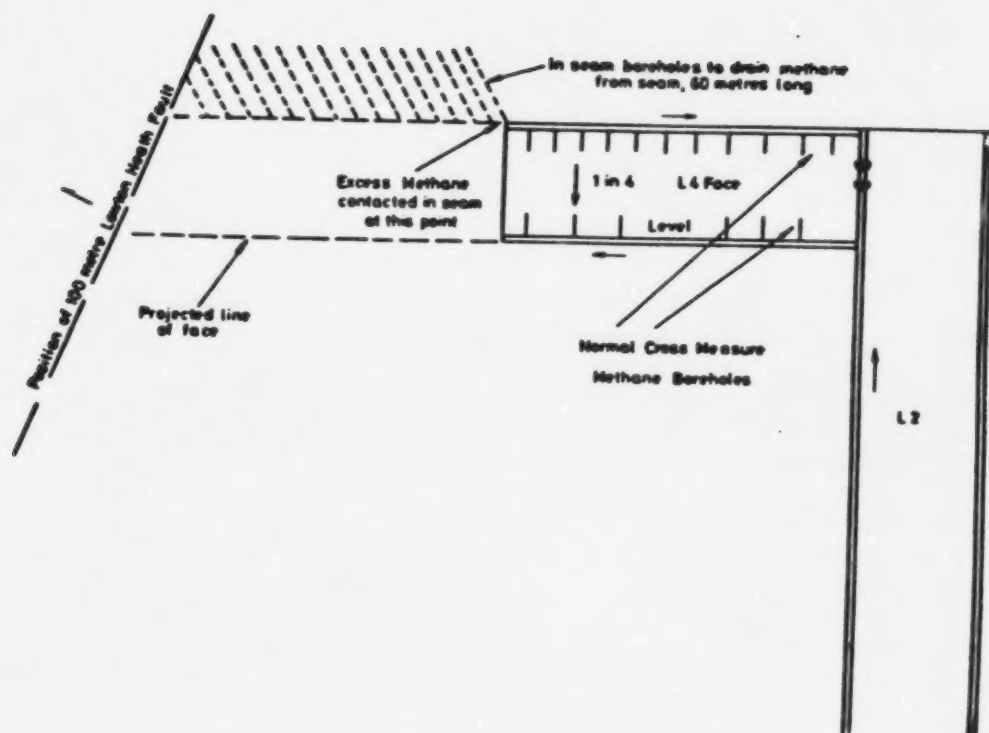


Figure 3-9
Golborne Colliery
Source: Highton (1982a)

3.3.3 SILVERDALE COLLIERY, WEST MIDLANDS

Seam:	Rowhurst
- Depth:	-
- Thickness:	-
Purpose:	Degassing of development drivages
Type of Drilling:	In-seam rotary in front of and at right angles to the drivages
Hydrofracturing:	None
Sources:	Highton (1982a)

.1 Relevance to Cape Breton Collieries

- Drilling program was considered a failure - the holes did not capture any significant amounts of gas.
- Permeability tests on the drillholes showed that the area of influence of adjacent holes was limited and the area affected by the very limited applied suction from each hole was negligible.
- As soon as fresh coal was exposed, gas was discharged into the roadway.
- It was decided that the most effective gas collector was the roadway itself.
- Rowhurst seam had been overworked by Winghay seam 100 m above and underworked in deeper seams 300 m below.
- It was considered that the effect of previous workings had changed gas conditions from an adsorbed to an absorbed state with the majority of gas being released by the previous workings. Gas in the Rowhurst seam was contained in minute breaks as within a sponge and any new exposure of coal quickly released further gas.
- A total of six 37 kw fans using twin 760 and 900 mm ducting delivering quantities in excess of 20 m³/s failed to keep a 300 m heading within statutory limits.

- Experience has shown that if a roadway was driven on the lower side of an existing roadway then that drivage effectively cut off the gas migration paths to the original roadway and gave an almost gas free condition. When designing the new layout with gas drainage roadways cognisance was taken in this phenomena. Shown in Figure 3-10. The gas roads were driven parallel to the main drivages and the method of work was such that the retreat face roadways were driven to the limit where they had to stop because of gas limitations. Using two gas drainage roads as shown in Figure 3-10, methane capture values of up to 300 L/s could be drained from the sealed areas. The drainage efficiency from the development area in total could vary between 45% and 55%.

Further controlled tests were carried out with in-seam gas drainage holes and with holes drilled at right angles from the roadway gas flows of up to 11 L/s pure were obtained with suctions of up to 2 kPa. When the methane is held in the absorbed state as at Silverdale, the majority of the gas is emitted in the first 50 metres of roadway from the newly exposed coal. To maximize the gas captured by drillholes, this zone would have to be saturated with holes; a condition which would mitigate against rapid rates of drivage advance. Under these conditions, it was considered that the gas drainage system was more effective.

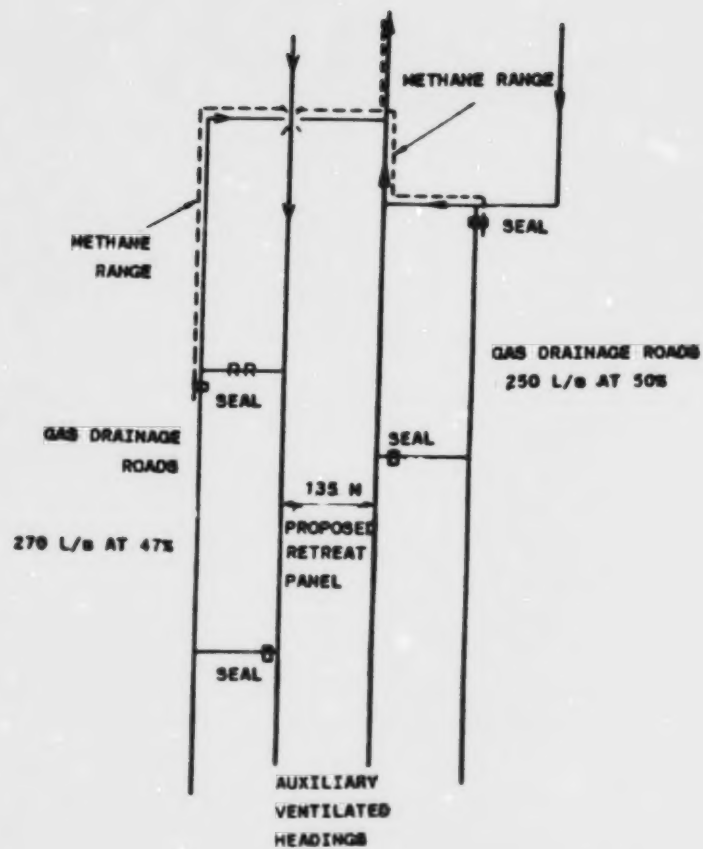


Figure 3-10
Silverdale Colliery
Source: Highton (1982a)

3.3.4 YORKSHIRE MAIN COLLIERY

Seam:	Barnsley
- Depth:	-
- Thickness:	-
Purpose:	-
Type of Drilling:	-
Hydrofracturing:	None
Sources:	Hayes (1982)

.1 Relevance to Cape Breton Collieries

- Measurements of gas makes from an advancing face practising methane drainage.
- 74% of gas in air at the outbye end of the district return originated between the statutory measuring points 10 m from the face in the intake and return.
- Of the gas measured, 48% came from the face, roof and floor and 52% from the goaf at the tail gate.
- Of the gas from the seam, 66% emanated from cleats 1.5 m apart.
- It was construed that 35% of district gas was from the advancing working face.

3.3.5 POINT OF AYR COLLIERY

Seam:	Durbog
- Depth:	500 m
- Thickness:	3.0 m (2.7 m mined)
Purpose:	Degassing development drivages
Type of Drilling:	50 mm rotary holes, steel standpipes 6 m long, 52 mm diameter in 75 mm hole. Standpipes sealed with quick setting cement
Hydrofracturing:	None
Sources:	Highton (1982a)

.1 Relevance to Cape Breton Collieries

- Before crushing some vitrain had granular texture. Friable fusain bands may provide degassing channels within the coal.
- In twin drivages in development headings, the rate of gas emission in the second heading would be dependent on its position relative to the first heading and the time interval between mining.
- A drainage pattern of boreholes (Figure 3-11) was repeated every 100 m.
- From out of the crosscut between the two headings, a long borehole was drilled in the seam for a distance of 200 m. The pillar between the two roadways was approximately 35 m and the methane hole was drilled in the centre of the pillar.
- From a small recess on the left-hand side of the No. 1 heading, three holes were drilled. These were 1) the first hole was in the seam, as near as possible parallel to the heading line for a distance of 100 m, 2) the second hole in the seam, at an angle of 45° to the line of heading for a distance of 70 m, and 3) the third hole in the seam at right angles to the roadway for a distance of 60 m.
- Out of the second drivage, three holes were drilled at the face of the drivage. These holes were on the right hand side of the heading. 1) the first hole drilled almost parallel to the proposed heading for a distance of 100 m, 2) the second hole drilled at 45° to the line of the heading for a distance of 70 m, and 3) the third hole drilled at right angles on the seam for a distance of 60 m.

This pattern of boreholes was repeated for each completed circuit of the development and was phased in with crosscuts at every 100 m interval.

- Based on past experience when working in virgin conditions, it was estimated that up to 500 L/s could be released in the solid headings when developing the first retreat panel. Therefore, with a maximum ventilation quantity of $12 \text{ m}^3/\text{s}$ in each heading and designing the system for a maximum methane content of 0.8% a total of 192 L/s could be diluted by the ventilation. The remaining 308 L/s would have to be captured by the drainage system. The boring programme was implemented and experience indicated that up to 30 holes were required on suction at any time to maintain the designed standards of ventilation. By the time the two headings approached the boundary, the methane captured by the drainage system was peaking at 645 L/s at a purity of 50% representing a flow of 327 L/s. The flow from individual boreholes varied from 50 L/s when initially drilled to a flow of as low as 5 L/s. To obtain optimum results from the holes the methane drainage system had to have sufficient capacity to create a suction of the inbye end of the hole.

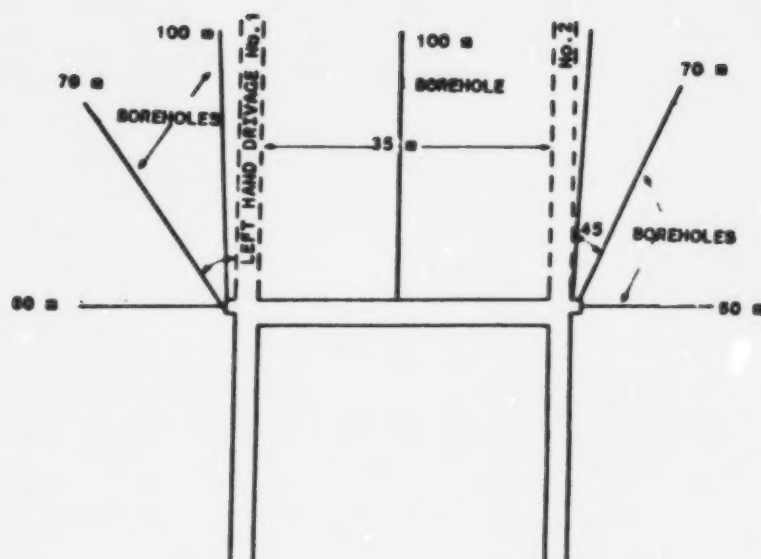


Figure 3-11
Drainage Scheme - Point of Ayr Colliery
Source: Highton (1982a)

3.4 UNITED STATES3.4.1 SUNNYSIDE NO. 1 MINE, UTAH

Seam:	Pittsburgh
- Depth:	760 m
- Thickness:	1.8 m
Purpose:	Methane drainage
Type of Drilling:	Electro hydraulic rotary
Hole Sizes:	89 mm bits
Hole Lengths:	131 m to 517 m
Drainage:	52 L/s to 111 L/s
Hydrofracturing:	From surface
Sources:	Finfinger et al (1982) Jones et al (1982)

.1 Relevance to Cape Breton Collieries

Long drillhole direction was uncontrolled and unpredictable.

Two test programs were carried out with a total of six holes being drilled. Holes Nos. 1 and 2, 89 mm in diameter were followed by holes 3, 4, 5, and 6 (Figure 3-12) Gas flows (Finfinger et al, 1982) were:

Hole No.	Depth (m)	Gas Flow L/s	
		Initial	Stabilized
1	131	52	20
2	137	42	25
3	238	14	3
4	517	60	38
5	412	111	49
6	307	74	41

During 9 months of study, production from holes No. 1 and 2 totalled 990,000 m³. After drilling No. 1 hole face flows declined from 76 to 41 L/s after 15 days. Hole No. 2 had little effect on methane face flows which remained steady over a period of 8 months.

Before the study was started, a vertical borehole was drilled and cased down to the 18 Dips sections to provide a means of transporting the methane from the boreholes to the surface. A polyethylene pipeline was installed from the underground drilling site to the bottom of the vertical borehole.

A continuous methane-sensing system, complete with automatic borehole shut-in valves, was installed to monitor for breaks in the underground pipelines. The sensing system was connected to methane monitors which were placed at 150 metre intervals along the length of the pipeline. The vertical borehole had flame arresters and lightning protection installed at the surface. The entire installation was checked at least once per shift by certified mine personnel.

Directional control of the drillholes in the horizontal plane was not attempted. The holes deviated to the right with the clockwise rotation with the exception of Hole No. 6 which deviated to the left even though all the drilling parameters were the same (thrust, rotational speed, bit design, and centralizer locations).

Jones et al (1982) quoted the following characteristics of the Sunnyside seam:

Rank	Geol. Era	Cover	Adsorbed Gas	Virgin Reservoir Pressure	Perm. (md)
High Vol. Bitum.	Cret.	2500 ft. 762 m	160 cu ft/ton 5 m ³ /t	620 psig 4272 kPa	2 (for Gilson Seam)

Average Gas Composition (Sunnyside) (MOL %)

CH ₄	C ₂ H ₆	C ₃ H ₈	O ₂	N ₂	Spec. Heat
99.01	0.18	0.03	0.30	0.48	1007 Btu/scf (37.5 MJ/m ³)

Comparison of flow predictions from a horizontal borehole and a fractured vertical well has been done. Jones et al (1982) used data of flows obtained in the Sunnyside project to compare with data obtained from flows in vertical wells where hydrofracturing had taken place. An attempt was made to predict flows that would likely be the result of stimulating the Sunnyside holes.

The basic comparison was between drainage from a 610 m (2000 ft) horizontal drillhole and a fractured well with the 'frac' size limited in order to assure roof stability.

Details of the analysis are presented in Section 5.

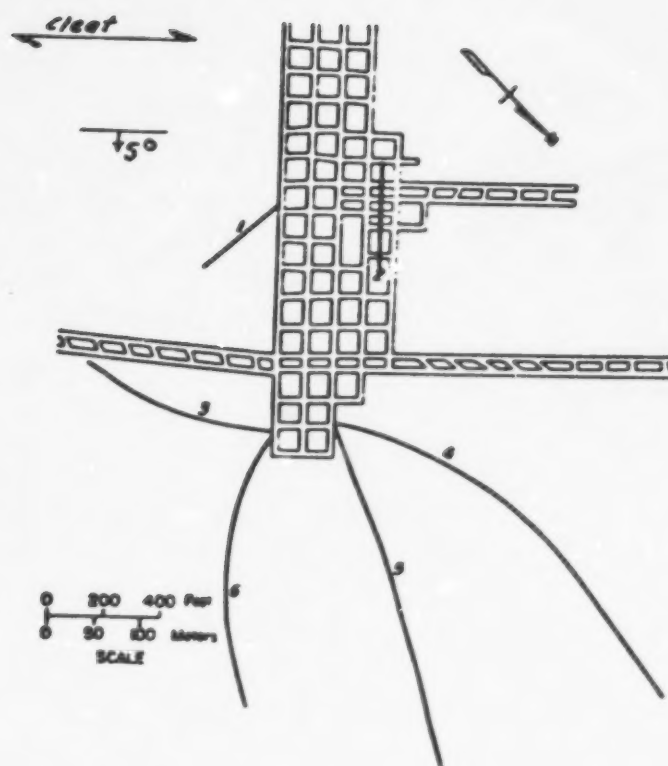


Figure 3-12
Sunnyside No. 1 Mine
Hole Locations - 18 Dips
Source: Finfinger et al (1982)

3.4.2 ISLAND CREEK COAL CO., V.P. NO. 5 MINE, VIRGINIA

Seam:	Pocahontas Nos. 3 and 4
- Depth:	-
- Thickness:	-
Purpose:	Coal seam drainage to dilute to less than 1% methane content.
Type of Drilling:	In-seam directional drilling Standpipes 102 mm (4 inches) diameter Length 6 to 18 m Rotary drilling
Drainage:	4 to 130 L/s per drillhole
Hydrofracturing:	None
Sources:	von Schonfeldt et al., 1980 Jones et al., 1982

.1 Relevance to Cape Breton Collieries

- Drilling rate was 30 m per shift.
- Hole control was by the number of stabilizers and thrust and rotational speed of the bit.

.2 Drainage Method

The drainage program included drilling both longholes within the coal seam and also cross measure holes into an overlying coal seam from the working seam. The drilling was one to several years ahead of mining to allow for drainage from the holes. Six holes were drilled within the Pocahontas No. 3 seam and two holes drilled cross measures up to and in the Pocahontas No. 4 seam, 17 m above. All holes were completed with 102 mm steel standpipes 6 to 18 m long grouted into place. The siting of the holes is shown in Figure 3-13.

Results from the drilling together with gas production are shown in Table 3-A. Gas analysis showed a methane content of 98%.

.3 In-Seam Drilling

The Pocahontas No. 3 coal bed is soft and friable so the bit had a tendency to ream downwards. This was counteracted by forcing the bit upwards allowing the hard roof to be a guiding horizon. Typical drilling results are shown in Figure 3-14. The basic tools used to control inclination were the number and spacing of centralizers, thrust and rotational speed of the bit. Frequent surveying of the hole was necessary until the proper centralizer configuration, bit thrust and speed was established. For the soft Pocahontas coal a Hughes Tool Blue Demon drag bit gave long bit life at an average drilling rate of 30 m per shift. For cross measure drilling 80 mm diamond plug bits were used.

.4 Methane Production

The holes gave a high emission for the first week or two after drilling. Production rates are shown on Table 3-A.

.5 Gas Content Measurements

The gas content was determined using 4.5 kg of fresh coal cuttings from the tail of the continuous miner and analyzed by desorbing over several hundred hours. The residual gas was determined by grinding, during which time the gas was measured and added to the desorbed gas.

The effect of drainage by holes 1, 2, and 3 (Figure 3-15) showed a substantial decrease in methane when the holes were intersected by mining 310 to 418 days after they were drilled (Figure 3-16). It indicated that the seam was approximately 80% degassed.

Desorption tests carried out on samples showed that degassed coal had a desorption rate 5% that of virgin coal over the same time interval (Figure 3-19).

.6 Methane Emission as a Function of Time

Jones et al 1982 notes that the Pocahontas No. 3 coal has a permeability of 2 to 9 md. Adsorbed gas is $13.4 \text{ m}^3/\text{t}$ and virgin reservoir pressure is 4500 kPa (650 psig) (for comparison, Russian permeabilities are noted as 0.003 to 0.05 md).

Unit methane liberation in m^3 per metre of rib showed that the daily emission rate decreased from a maximum of 14 m^3 at the face to 2 m^3 at a distance of 610 m from the face. This data indicated that methane emissions into entries could be reduced by 20% or less of the maximum emission rate if suitable drainage was maintained. A minimum of 40 weeks drainage time would be required for total drainage with 15 weeks providing for 50% emission.

Hole Number	Depth (m)	Unit Production Range (L/s/m)	Maximum Pressure Recorded (kg/cm ²)	Life Of Hole (days)	Average Daily Production (m ³)	Length Of Standpipe (m)
Oxy #1	152	0.4 - 0.01	4.2	418	301	6
Oxy #2	153	0.46 - 0.01	3.7	310	673	6
Oxy #3	153	0.47 - 0.03	3.5	323	1306	6
Oxy #4	154 (84 in coal)	0.39 - 0.16 (4.20 - 1.77)	3.5	126	1478	6
Oxy #5	475	0.35 - 0.06	3.5	153	6296	15
Oxy #6	525	0.63 - 0.17	5.6	167	11210	12
Oxy #7	117 (11 in coal)	0.93 - 0.38	17.6	16	346	4.6
Oxy #8	201 (35 in coal)	0.47 - 0.30	12.4	14	1397	4.6

TABLE 3-A
V.P. No. 5 Mine - Drainage Data
After: von Schonfeldt et al (1980)

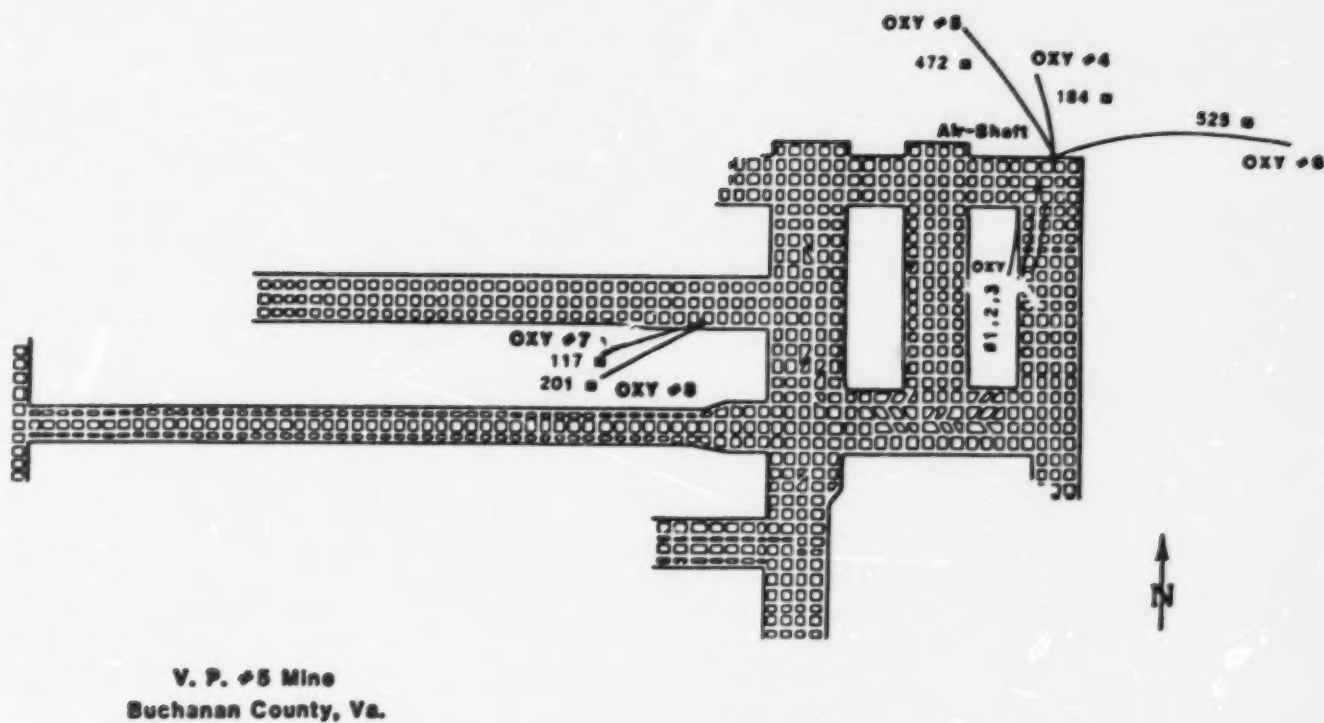


Figure 3-13
Location of Methane Drainage Holes
Source: von Schonfeldt et al (1980)

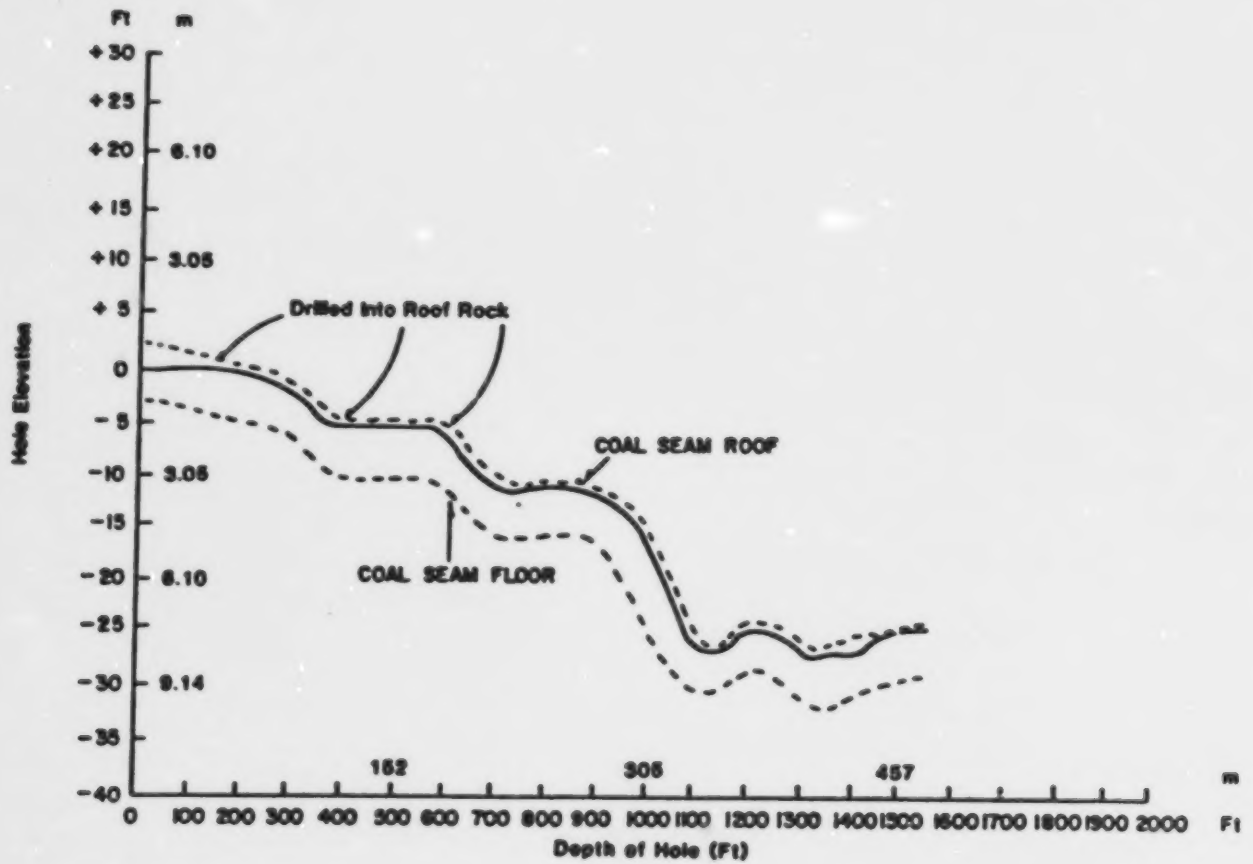


Figure 3-14
Trajectory of Oxy No. 5 Hole
Source: von Schonfeldt et al (1980)

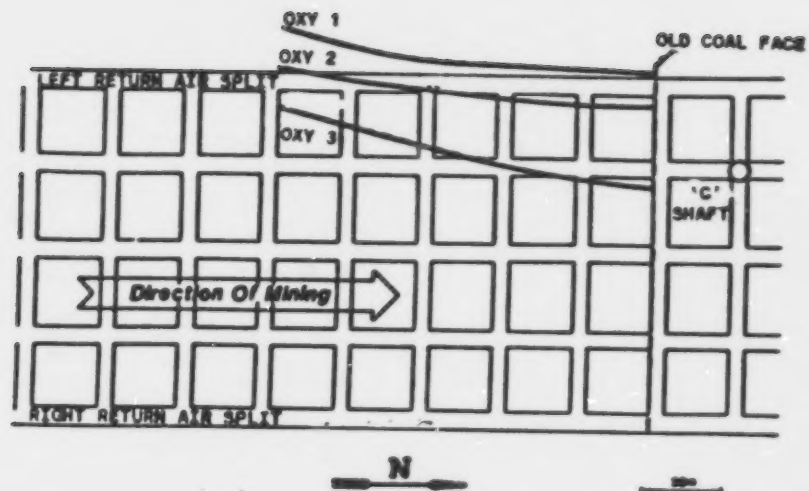


Figure 3-15

Mine Plan and Location of Intercepted Holes

Source: von Schonfeldt et al (1980)

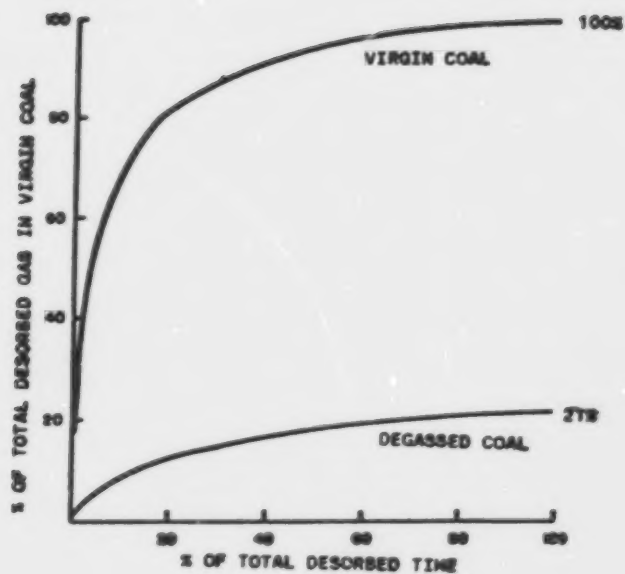


Figure 3-16
Desorption Curves for Virgin and Degassed Coal Samples
Source: von Schonfeldt et al (1980)

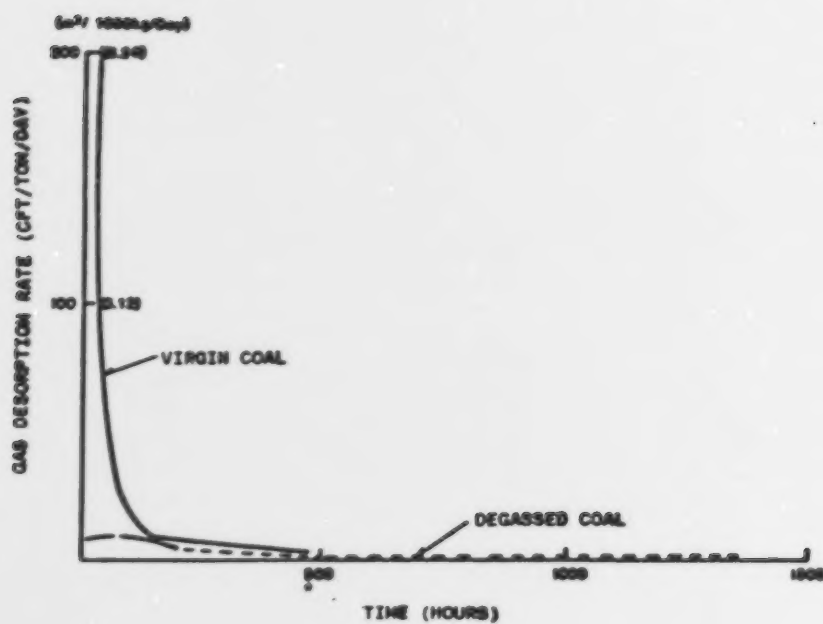


Figure 3-17
Effect of Inseam Drainage on Gas Desorption Rate
Source: von Schonfeldt et al (1980)

3.4.3 SOLDIER CANYON MINE, CARBON COUNTY, UTAH

Seam:	Rock Canyon
Depth:	475 m to 498 m (1560 to 1636 ft)
Thickness:	Minimum 1.8 m (6 ft)
Purpose of Drilling:	Extraction of coalbed methane for sale
Type of Drilling:	- Initially rotary then down hole motor. - J.H. Fletcher and Co. Model LHD-13 drilling system and Navi-Drill down hole hydraulic motor. - J.H. Fletcher telemetry survey tool. - Standpipe 15 cm (6 in) diameter, 15 m (50 ft) long grouted in coal seam - Length of holes - 100 to 700 m (328 to 2296 ft)
Drainage:	From right angles to butt cleat 0.5 L/s Mananna Mine - maximum 35 L/s, average 20 L/s. Water flow 0 to 0.7 L/s
Hydrofracturing:	None
Source:	Schwoebel, 1987

.1 Relevance to Cape Breton Collieries

- Downhole motor drilling preferred for long holes.
- Holes at right angles to face cleat produced most gas.
- Vacuum pumps significantly assisted drainage.
- Back pressure at face reduced gas flow rates.
- Coalbed methane production is highly dependent on effective water removal.
- Dip holes drained of water by using compressed air. Frequent drainage is required.
- Gas-water separation devices are installed at the hole standpipes.
- Methane transported underground through thick walled low-pressure (830 kPa or 120 psi) polyethylene transmission pipe. 6 m (20 ft) sections of pipe are installed using MSHA -approved fusion techniques.
- A thin walled 13 mm (0.5 in) polyethylene pipe accompanies the transmission lines containing compressed air. The tubing is connected to pneumatic valves and shuts in each borehole. The

thin wall tubing is likely to fail before the main tubing thus preventing methane to discharge into the roadway should the line be damaged.

- Methane detectors are also placed along the roadway at approximately 760 m (2500 ft) intervals. These also are interfaced to electrically-actuated, three-way dump valves attached to the 13 mm tubing. The dump valves are set to release air pressure in the tubing when methane concentration in the airway reached 1.5%. All holes are then closed.
- Generation and control of the compressed air is on the surface.

3.5 CANADA

3.5.1 NO. 26 COLLIERY, CAPE BRETON

Seam:	Harbour
- Depth:	750 m
- Section:	2.0 m
Purpose:	Test for methane gas production
Type of Drilling:	Turmag Drill Rig
	Hole Diameter 100 mm
	Hole Length 32 to 70 m
Drainage:	0.03 to 0.18 L/s
Source:	CBDC report, March 1982

.1 In-Seam Drilling of No. 26 Colliery

Excerpt from progress report during the joint venture by CBDC and Novacorp dated March 1982, in the test program to measure methane flows from boreholes drilled in the Harbour Seam (The mine closed in 1984 following a major fire underground).

The test site was at the bottom of the belt deep in the Harbour Seam undersea workings approximately 6 km north of the shoreline at a depth of 750 m below the sea bed. The design of the test was to drill in coal a distance of 122 m using a Turmag drill rig with a 100 mm bit, and to measure the gas pressures and flows from the holes.

The coal seam was dipping at approximately 7° and the intent was to start the holes 0.76 m from the roof of the seam and to stay within the seam. It was thought that abutment pressures would extend 60 m (200 ft) in advance of the working and that 122 m plus should clear the abutment zone.

Table 3-B shows a summary of the results achieved. The drillers had difficulty in controlling the horizon of the drill holes and each one entered the roof above the seam resulting in difficult drilling.

None of the holes reached the design length, with total lengths ranging between 27 m and 70 m. The longest distance within coal was 48.8 m. With the difficult drilling, it was assumed that the holes did not clear the high pressure of the abutment zone. Measured flows of methane varied from 0.03 to 0.18 L/s. A maximum reading of 0.26 L/s from one hole was reported in December 1981. No attempts were made to hydrofracture either the coal or the roof. At the same time of the tests, the production of methane within the mine averaged 14,170 m³ (500,000 cu ft) from the north side and 12,755 m³ (450,000 cu ft) from the south side. It should be noted that in 1983 and 1984 two major gas and rock outbursts occurred in the main deeps drivages (Aston and Cain, 1985) in which there was a "release of large quantities of methane while the ejected material consisted of pulverized fine/medium-grained sandstone.

All outbursts occurred where an overlying sandstone unit came within 3 m of the top of the coal seam and were initiated by shot firing. The depth of the outburst prone zone was below 700 m.

Drill Hole No.	Collar Dip	From Top of Seam ft.	From Top of Seam m	Total Length of Borehole ft.	Total Length of Borehole m	Stone Position* ft.	Stone Position* m	Uncased Coal ft.	Uncased Coal m	Volume of CH ₄ /Day		Volume of CH ₄ /Day per Unit Length in Coal ft. ³ /ft	Volume of CH ₄ /Day per Unit Length in Coal m ³ /m
										ft. ³	m ³		
1	-7.50	2.5	0.76	130	40	94	29	83	25	214	6.1	2.58	0.24
2	-9.00	2.5	0.76	87	27	87	26.5	67	20.4	COULD	NOT	MEASURE	
3	-5.50	2.5	0.76	105	32	87	26.5	67	20.4	98	2.7	1.46	0.13
4	-3.00	2.5	0.76	202	61	177	56	160	48.8	247	7.0	1.54	0.14
5	-3.00	2.5	0.76	152	46	115	35	105	32.0	124	3.5	1.18	0.11
6	0.00	1.5	0.45	231	70	40	12	102	31.0	540	15.3	5.29	0.49

* Stone position is the point where stone was first encountered in the hole.

Table 3-B

No. 26 Colliery - Summary of Predrainage Holes
(Drilled in the Harbour Seam at the Bottom of the Belt Deepa)
Source: CBDC, Internal Report, 1982

PRE-DRAINAGE IN COAL MINES

SECTION 4 - IN-SEAM DRILLING

TABLE OF CONTENTS

4.1	INTRODUCTION	4-1
4.2	DRILLING METHODS	4-1
4.3	DRILLING EQUIPMENT	4-2
.1	General	4-2
.2	Bits	4-2
4.4	DRILL HOLE COLLARS	4-3
.1	General	4-3
.2	Use of Non-Metal Collars	4-5
.3	Blowout Preventor and Shut-In Valve	4-6
4.5	DIRECTIONAL CONTROL	4-6
.1	Introduction	4-6
.2	Survey Tools	4-7
.3	Rotary	4-8
.4	Downhole Motor	4-9
.5	Case History - Phalen Mine	4-10

LIST OF FIGURES

4-1	Arrangement of Stuffing box and Separator Tank
4-2	Location of Pneumatic Shutoff Valve in Methane Drainage Pipeline System
4-3	Natural Gamma Log of Hole
4-4	Layout of Surveying Equipment
4-5	Gamma-ray Seam Calibration
4-6	Fault Displacement Estimate
4-7	Positions for Centralizers on the Drill String
4-8	Upwardly Cocked Drill Assembly
4-9	Downhole Rotor and Stator Motor
4-10	Downhole Turbine Motor
4-11	Navigation Sub with DTU Housing

PRE-DRAINAGE IN COAL MINES

4.0 IN-SEAM DRILLING

4.1 INTRODUCTION

Drilling to extract methane from coal seams has been carried out from both the surface and underground mine workings.

Longhole, directional drilling has been carried out in the mining and petroleum industries for many years. The relatively recent development of downhole motors has greatly improved the directional control of drill holes.

Drilling from the surface can utilize large equipment, but requires longer holes; underground drilling is constrained by space and ventilation, but drill sites are in close proximity to most targets. Cape Breton Collieries do not have the luxury of choice, being undersea operations any drilling will have to be carried out in the mine.

4.2 DRILLING METHODS

The case histories (Section 3) have shown that the selection of a drilling method is dependent on the type of pre-drainage required. Often locally available equipment using small diameter drill strings have been deployed where in-seam gas effected the mining operations in specific locations. Few longhole drilling projects have been attempted underground (Table 4-1). Successful drainage has been accomplished by drilling patterns of short small diameter holes (Figure 3-5).

Conventional rotary or downhole motor equipment can be used from surface or underground for directional drilling to and within the coal seam.

4.3 DRILLING EQUIPMENT

.1 General

Drill equipment must be chosen to meet the following criteria:

- When drilling in underground coal mines, all equipment must meet flame-proof and all other safety requirements.
- Air is not to be used as a circulating medium as incendiary sparks could ignite a methane/air explosive mixture.
- Provide means of shutting off hole with drill rods in place.
- Provide means of abstracting gas and water separately whilst shut off for drilling.
- Provide regular dip and azimuth measurements.
- Provide means of deflecting hole direction.
- Allow for insertion of sonde into drill string to determine rock type at or near bit during drilling.

.2 Bits

When deciding on the selection of bits, the size and depth of hole together with the composition of the material to be drilled must be considered.

.1 Coal Augers

For soft and brittle coal seams, a simple flight auger with cutting head can be used for short holes. Coal production augers with diameters from 0.5 m to over 2.0 m can be used for making holes up to 75 m in length although the practical limitation for underground is 40 to 50 m and then only in non-gassy seams. Coal augers can only be used for short, free-flow, open holes and not for a controlled test program. Auger flights do not permit directional drilling and are not suitable for hard materials.

.2 Drag, Three Cone Roller and Plug Bits

The main types of functional drill bits are three blade drag bit, three cone roller bit and the plug bit. Penetration rates by the three bits in tests in Ohio coal conducted by USBM (Cervik et al, 1975) were:

BIT TYPE	PENETRATION RATE	
	ft/min	m/min
Drag	11 - 12	3.4 - 3.7
Three Cone Roller	3 - 4	0.9 - 1.2
Plug	2 - 5	0.6 - 1.5

For fixed levels of bit thrust and rotational speed in coal, the drag bit has the highest penetration rate and is the least costly, but the bit will not penetrate hard strata.

Drag bits were used efficiently in longhole, in-seam drill holes in the Jim Walter Resources No. 4 Mine, one hole being 307 m in length. (Perry et al, 1982).

4.4 DRILL HOLE COLLARS

.1 General

To ensure that gas encountered by the drill hole can be controlled, it is essential that the hole be sealed at the interface with the workings. Depending on the permeability and fracturing of the coal and strata in the vicinity of the drill hole, there is likely to be significant leakage into workings especially if negative pressure exhaust ventilation is used.

To prevent such leakage, a pipe with an internal diameter slightly larger than that of the drill hole and of sufficient length to clear a leakage zone is grouted into the drill hole site. If the hole collar is to

be mined later, the pipe must be of material which will not damage a mining machine (e.g. a plastic pipe should be used).

The length of pipe necessary can be ascertained by air injection tests or initially, the 6 m length used by USBM can be tried. The pipe forms a collar, or standpipe, for the longhole.

It is advisable that a hole slightly larger than the pipe be drilled a metre longer than the pipe and the pipe be cement grouted within the hole. When the grout is set, the collar and gas control valves can be tested up to design pressure and remain under pressure for 24 hours. If satisfactory, the longhole can be drilled through the pipe and grout into the coalbed. In the case of a friable and broken coal, it may prove necessary to install the collar in a horizon above or below that which is in the coal horizon.

The USBM have developed a standard procedure for collaring holes (Finfinger et al, 1982). First a 15 cm hole is drilled to a depth of 6 m. Next a 6.1 m length of 10 cm schedule 40 pipe is grouted into place. When grouting is completed, a 10 cm full-throated valve and specially designed stuffing box is attached to the pipe. The stuffing box is designed to separate the gas from the water and drill cuttings during drilling (Figure 4-1). Flexible wire-reinforced duct tubing is then connected to the top of the stuffing box and is connected to a pipeline for transporting methane to either a return airway or to a pipe for venting methane to the surface.

During the drilling phase, methane concentrations are monitored every 20 minutes with hand held methanometers. Continuous reading methane sensing devices are connected to automatic borehole shut-in valves after drilling has been completed.

.2 Use Of Non-Metal Collars

There is concern that plastic collars or liners will be subject to static electricity which could ignite an explosive methane/air mixture. Plastic stand pipes, 10 m long, were installed on in-seam drillholes from previously developed gate roads in front of a retreat longwall face in West Germany (Noack, 1982). The annular space between the plastic tube and the borehole was sealed by rubber sleeves and filled partially with polyurethane foam. Borehole diameters were 45, 65 and 95 mm, maximum length of boreholes was 120 m and distances between the boreholes were 5, 10 and 15 m. The discussion appended to Noack's (1982) paper raised the concerns of static electricity build-up.

In reply to the question "Have the plastic pipes used in stand pipes and also in the roadways been approved by the Mine Departments and are they in fact, anti-static?"

Reply by K. Noack (Westfälische Berggewerkschaftskasse, West Germany):

"In Germany there are two different qualities of plastic tubes which are approved to be used in the strata, but none to be used as methane drainage lines in the roadways. These two different qualities are anti-static. One has a coating of anti-static material which must remain undamaged. Therefore, drilling through this sort of stand pipe is not allowed and in this case, the borehole must be drilled to its whole length before setting the stand pipe. The other type of tube is fully anti-static and there are no such constrictions as with the first one. The tubes are delivered by the firms Korfman in Witten and Muller and Borggrafe in Gevelsberg."

Comments by R. King (U.S. Bureau of Mines):

"The plastic pipes are not approved for static electricity by the U.S. Mines Safety and Health Administration which is the regulatory body. They do recognize in the ventilation plans that the mine operators will make some type of static electricity check. In other words, it is ignored; that is

exactly what happens. In reality, it is ignored pretty much and so there has only been one steel pipeline (which was illustrated) and the rest have been plastic."

A second question was "When the boreholes are closed off, does the pressure build up to the seam gas pressure or does it leak past the stand pipe into the airway, thus giving pollution of gas in that particular airway?" K. Noack replied that there is a pressure build up in shut-in holes, but there is not much additional flow into the roadways because of the tightness of the seals. Pressure does not reach a dangerous level in normal cases, but probably a dangerous pressure could build up in outburst prone zones. R. King (U.S. Bureau of Mines) commented that stand pipes are kept as long as possible and are grouted in as well because emissions will occur around that pipe.

.3 Blowout Preventor And Shut-In Valve

It is essential that provision is made for a manually operated blowout preventor, or stuffing box, to control high methane or water pressures. For many years, the U.K. "Burnside" stuffing box was used, but in the 1970's the USBM designed a box which permitted control of gas or water and also removal of cuttings. This machine is attached to the drill hole collar. Within the rods, a one-way check valve prevents gas or water blowing back down the rods. This check valve operates when drill pipe is being pulled or hole surveys are being made. A control valve coupling is supplied to allow either suction or pressure to be applied at the drill hole collar (Figure 4-2).

4.5 DIRECTIONAL CONTROL

.1 Introduction

In-seam drilling requires the capability to vary and control the drill bit direction during the drilling process. The mining and petroleum

industries have development techniques to successfully intersect specific target locations with a drill bit. The additional challenge with in-seam drilling is to stay within a coal seam while still reaching a particular target or maintaining a specified direction.

Survey tools, currently available, can be used to determine the position of the bit relative to the collar. This position does not give the bit position relative to the roof or floor of the coal seam.

.2 Survey Tools

Survey tools used by the USBM are instruments such as the Sperry-Sun or Eastman Well Surveying Co. single-shot survey instruments. Cervik et al (1975) describe the essential elements of the single-shot surveying instrument as the inclination unit, which is in the form of an inverted plumb bob, batteries to furnish electrical power, and a timer, which controls the electrical circuits and illuminates the angle unit at a preset time to record the inclination on a film disk. Special loading and developing tanks permit handling of the photographic record disk in daylight or without turning off cap lamps underground.

The instrument records hole inclination and azimuth by photographing an inclinometer and a magnetic compass. The instrument is designed to be pumped to the end of the rods using water. Surveys are usually taken at 6 to 9 m intervals. At a hole depth of approximately 300 m, one survey shot takes about 1 hour.

One case study (Perry et al, 1982) showed where an in-seam drill hole was later intersected by workings in the seam, there was a difference in survey of approximately 3° azimuth giving a displacement error of 13.7 m (45 ft). A possible explanation could be magnetic interference.

Natural gamma ray logs have been used extensively to identify coal seams in surface drillholes. Thompson (1984) reported on an in-seam

drilling trial from surface that used this logging tool to determine whether the drill string was in or out of the coal seam (Figure 4-3).

British Coal (1987 and Hextall, 1983) have been developing a downhole survey tool to give the position of the bit relative to the drillhole collar and to the seam roof and floor. This tool was designed (Figure 4-4);

- to be pumped into NQ rods,
- to relay back to the collar, information on rod inclination and natural gamma ray counts,
- to transmit the information via the wireline cable and winch,
- to display the information on a LCD readout unit,
- to be intrinsically safe.

The natural gamma radiation of the strata is measured both up and down giving the relative position of the tool in the seam; detailed calibration of each coal seam is required (Figure 4-5). To date, British Coal has completed five field trials in exploration and fault definition (Figure 4-6).

.3 Rotary

Drill strings can be used in rotary drilling to control both torque and bit pressure. Directional control of the drill string can also be effected by the positioning of centralizers at pre-determined intervals relative to the drill bit (Figure 4-7). Once the direction of the bit has been established by using the survey instrument, a centralizer placed directly behind the bit can be used to change direction (Perry et al, 1982). Figure 4-8 illustrates an example where the centralizer cocked the drilling assembly (bit, centralizer and BQ rod) about 0°-54° upward. The corrective measures can only be taken to keep the hole in the coal seam when the position of the bit in relation to the seam has been identified.

Each case requires knowledge of the seam itself, thickness, position and compaction of dirt bands, composition of roof and floor, including hardness in relation to coal, direction of cleat, frequency of jointing and probability of faulting, washouts and intrusive dykes. One option is for continuous coring and geological interpretation of core logs. This requires expert local knowledge by an experienced geologist. Interpretation may be augmented by the use of an in-hole television camera (Oyler and Diamond, 1982).

In the case of a soft coal with hard floor, the bit is likely to stay within the coal and can be "bounced off" the floor. Provided the seam is thick enough, the hole will be in coal for methane drainage purposes.

.4 Downhole Motor

Downhole motors deliver drilling rotation directly to the bit either by rotor and stator (Figure 4-9) or by turbine (Figure 4-10).

The motor can be fitted ahead of an integral string stabilizer. For directional drilling, a Double-Tilted U-Joint Housing can be fitted before the Standard Bearing Assembly (with an Integral Stabilizer and the bit (Figure 4-11). Norton Christensen (1984) state that in principle, the bit axis will be tilted relative to the borehole axis. The position of at least two stabilizers on the tool, the bit position and the bit offset precisely define a radius of a circle. The use of two stabilizers has three important purposes:

- .1 The stabilizers are used as tangency points to define the radius of the circle.
- .2 The bearing stabilizer maintains the bit tilt angle and minimizes any deflections which might increase or decrease the angle.

- .3 The top stabilizer can be used to decrease the rate of built by placing it further up the string.

Figure 4-9 shows the principle of the navigation tool with the double tilted U-joint housing. This system can be used with every type of bearing assembly. The geometry is identical to that of the borehole axis and the axis of the drive shaft lies in the center of the bottom stabilizer. The motor section is in a concentric position thus avoiding a severe tool bending when tripping in the hole or when rotating the string.

With the larger holes (generally 100 mm (4") diameter), the larger survey instruments can be used. A summary of case studies using downhole motors with regard to coal is shown in Table 4-2.

In the hard rock mines, compressed air driven in-the-hole drills such as the Inco GO-60 I.T.H. (CMS-CD-90) drill are being fitted with computer operated directional drilling (Kelly and England, 1987). At the present time, these hard rock drills are too large for use in coal mines, but developments should be monitored as to possible adaption and technology transfer to coal.

.5 Case History - Phalen Mine

A test drilling of a longhole using a downhole motor was made in the Phalen Colliery in 1985.

• Purpose

The hole was drilled with the intent to test for old workings of an adjacent coal mine from the last century. The old workings could have extended closer to the Phalen Deeps than the mine plan showed.

The Phalen coal seam is 1.95 m (6.4 ft) thick and bedded in on both sides by a mudstone. The seam dips 16° to the north underneath the Atlantic Ocean. The hole was to be drilled in a N 59° E direction (dipping 90°).

• Equipment Comprised

Drill:	Electric Longyear 44.
Pump Unit:	2 Bean 435 Connected with a manifold.
In-Hole Motor:	Boyles Bros. 60.3 mm (2 3/8 in.) size Navi Motor.
Deflection	
Device:	Deflection Shoe (Conoco Shoe)
Bit:	Norton Christensen Stratapax size 75.7 mm (2.980 in.).
Rodstring:	BQ Wire line. Behind the Navi Motor was added 3 x 3.05 m (10 ft) non-magnetic B-size rods (BeCu) to avoid magnetic interference from the Navi motor or rodstring.
Survey	
Instrument:	Eastman singleshot Type E. The survey rods were arranged so the survey instruments would be placed 4.5 m (15 ft) behind the Navi motor in the middle of the 9.1 m (30 ft) non-magnetic rods.

The drilling plan was to commence the hole 0.6 m (2 ft) below the coal seam and to dip at -5° until the coal seam was intersected, which would be at approximately 10 m (30 ft). The intent was then to continue within the coal seam for 200 m (650 ft).

The hole was actually drilled 175 m (573 ft), but difficulties were encountered in keeping the motor within the seam and in getting repeatable survey results in 9 m of non-magnetic rods. Opinion on the results of the test (Personal communication S. Forgeron) was: surveying of the hole was most critical and drilling with this equipment should result in 30 m to 60 m (100 ft to 200 ft) within the seam for each turn of direction.

EXPERIENCE WITH HORIZONTAL DRILLING
(U.S.A.)

TABLE 4-1

Source: Finfinger and Cervic, 1980

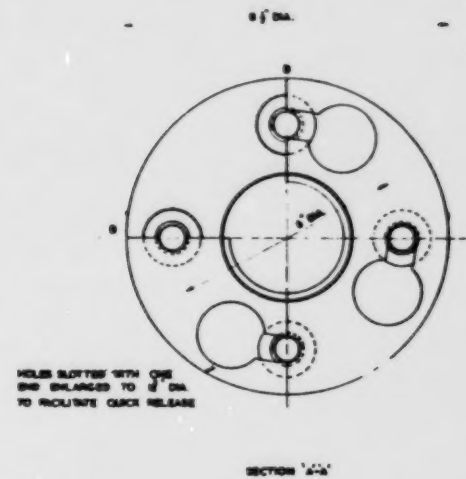
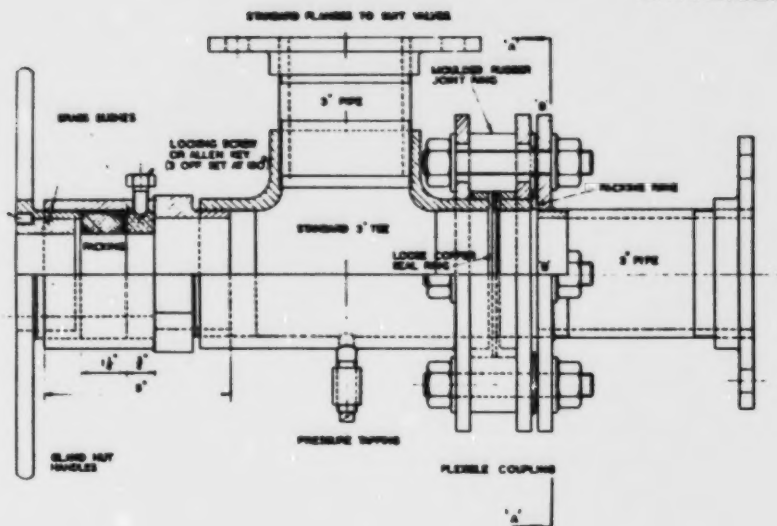
Mine	Seam	Total Length Drilled m	Longest Single Hole m	
Bethlehem Mines Corp. Marianna No. 58	Pittsburgh	1,740	760	
Consol	Pittsburgh	6,100	300	
Eastern Asso- ciated Coal	Pittsburgh	Multiple	300	Longyear Electrohydraulic
Kaiser Steel	Sunnyside	300	150	
Maple Mining Co.	Beckley	1,480	185	
Jim Walters	Mary Lee and Blue Creek	Multiple	300	
Island Creek	Pocahontas	Multiple	300	Acker Drill
Kerr-McGee	Hartshorne	2 Holes	1,000	
Mid-Continent Coal & Coke Corp	B Seam		45	EDECO Minihydrac 50 mm on 15 m centres
United States Steel Corp Oak Grove	Blue Creek	15 Holes	317	Joy Face Type Drill modified by Acker Drill Co.

SUMMARY OF IN-HOLE MOTOR CASE STUDIES
TABLE 4-2

Country	Drill	Hole Diam (mm)	Hole Length (m)	Deflection Device	Survey Instrument	Source
Canada CBDC	Electric Longyear Boyles Bros Navi Motor	75.7	175	Conoco Shoe	Eastman Single Shot	Edling, 1985
U.S.A. Emerald Mine	Dyna-Drill	76	504	Muli Shoe	Single Shot plus TV Camera	Oyler and Diamond, 1982 Surface Drilling All costs provided.
U.S.A. Pittsburgh Seam	Conoco/ Fletcher	75 to 100	to 600	Conoco Deflection Device	Conoco Instrument	Thakur and Poundstone, 1980
Australia ACIRL	N/A		825		Photographic Survey tool	Hebblewhite et al, 1985
U.K. Cotgrave 1983 Cotgrave 1986 Florence 1983/84 Agecroft 1984 Lee Hall 1985	Hydraq Ltd. MK II N/A	90	250 172 150 190 400		Salford Electrical Instruments Ltd.	British Coal Headquarters Technical Department, Guided longhole drilling.

AR-124

ARRANGEMENT OF STUFFING BOX



SEPARATOR TANK

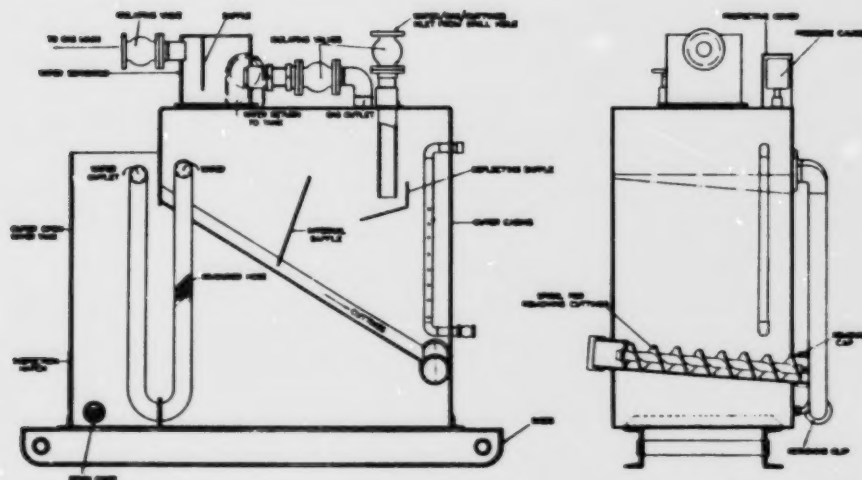


Figure 4-1
Arrangement of Stuffing Box and Separator Tank
Source: Dames and Moore, 1977

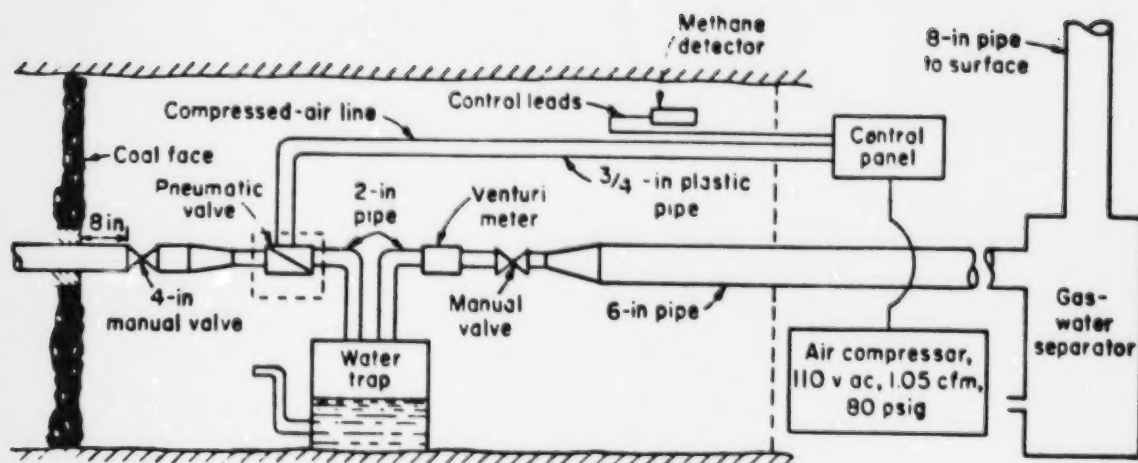


Figure 4-2
Location of Pneumatic Shutoff Valve in
Methane Drainage Pipeline System
 Source: Prosser et al., 1981

AR-124

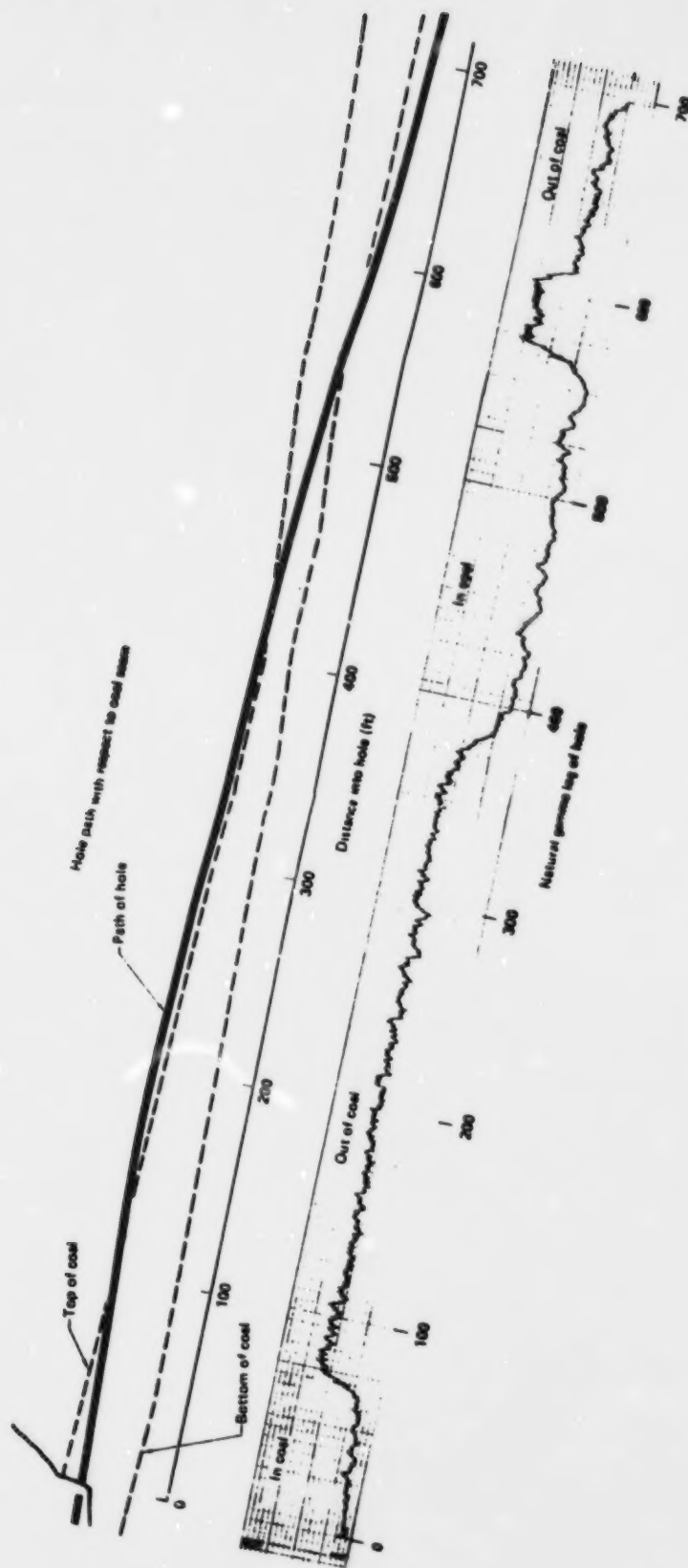


Figure 4-3
Natural Gamma Log of Hole
Source: Thompson, 1984

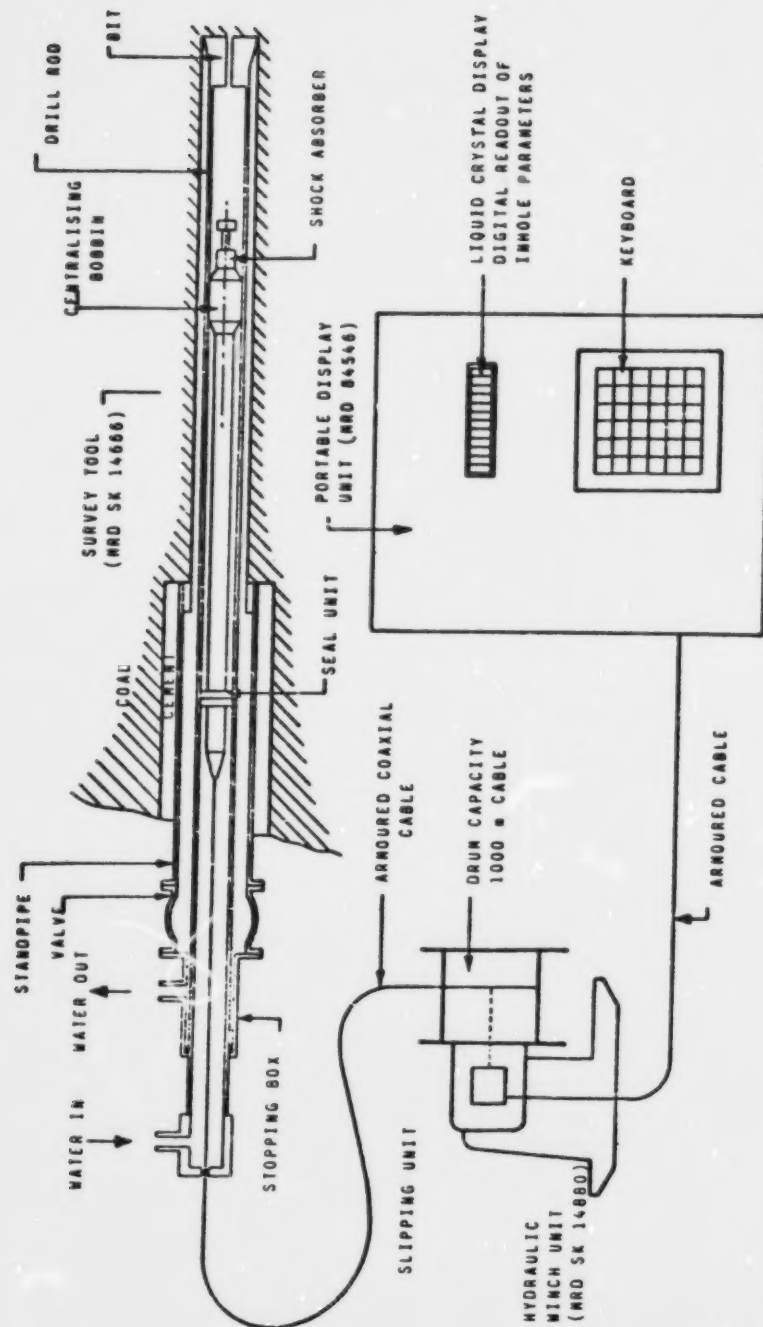


Figure 4-4
Layout Of Surveying Equipment, British Coal
Source: Hextall, 1983

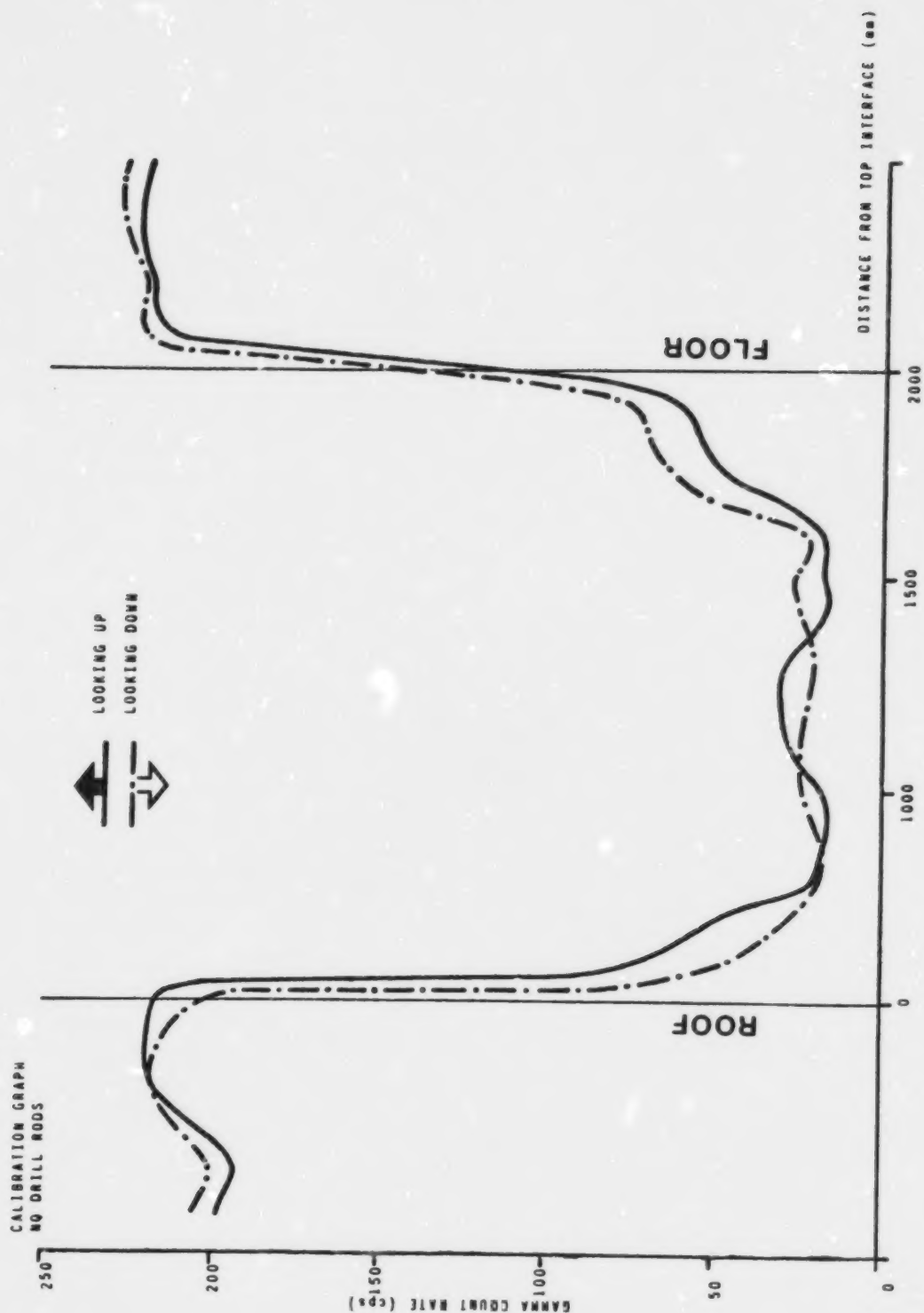


Figure 4-5
Gamma ray Seam Calibration
Deep Hard Seam - Cotgrave Colliery
Source: Hextall, 1983

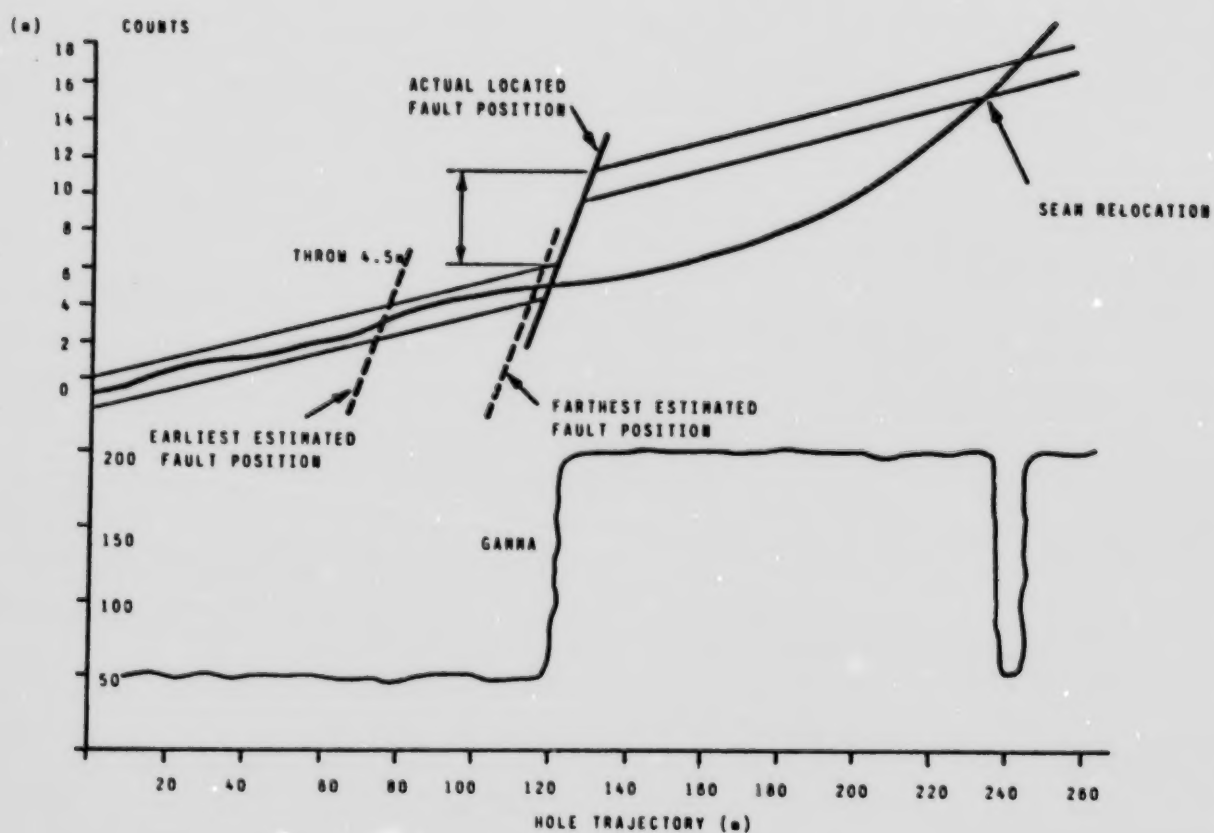


Figure 4-6
 Fault Displacement Estimate
 Cotgrave Underground Trial H60's (June 83)
 Source: Hextall, 1983

AR-124

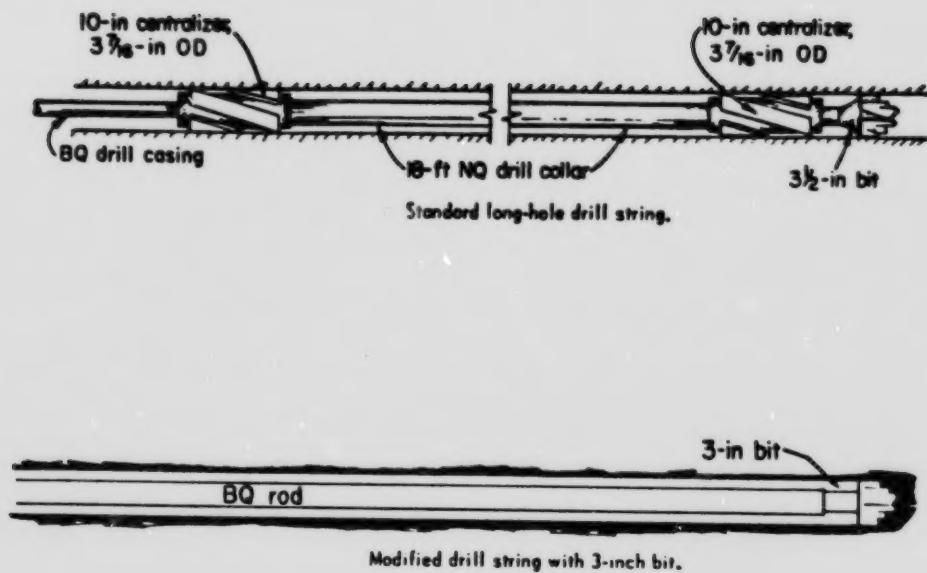


Figure 4-7
Positions for Centralizers on the Drill String
Source: Perry et al., 1982

AR-124

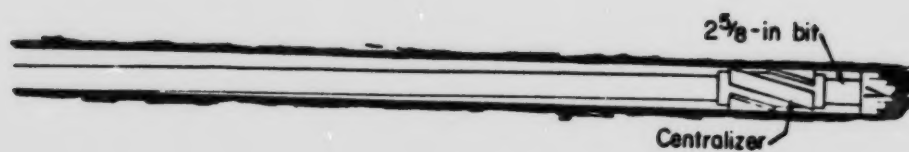


Figure 4-8
Upwardly Cocked Drill Assembly
Source: Perry et al., 1982

Navi-Drill[®] Mach 3

Benefits

- Precise directional control
- Motor drilling with low rig mud flow capacity
- Versatile performance with a variety of diamond bit types
- Wide rotation speed range
- Reduced wear on drill string

Features

A (1/2) lobe rotor/stator configuration allows the Mach 3 to deliver optimum rotation speed and torque with low mud flow rates in directional or straight-interval applications. Pressure drops between 350 and 1500 psi at the bit provide effective operation, making the Navi-Drill Mach 3 particularly well suited to rigs with moderate pump capacity.

Applications

A wide range of rotation speeds are possible with the Mach 3 Navi-Drill by varying mud flow rates. This offers versatile performance in directional control and short run applications such as:

- Rathole and mousehole drilling
- Controlled straight-interval drilling
- Production/workover operations

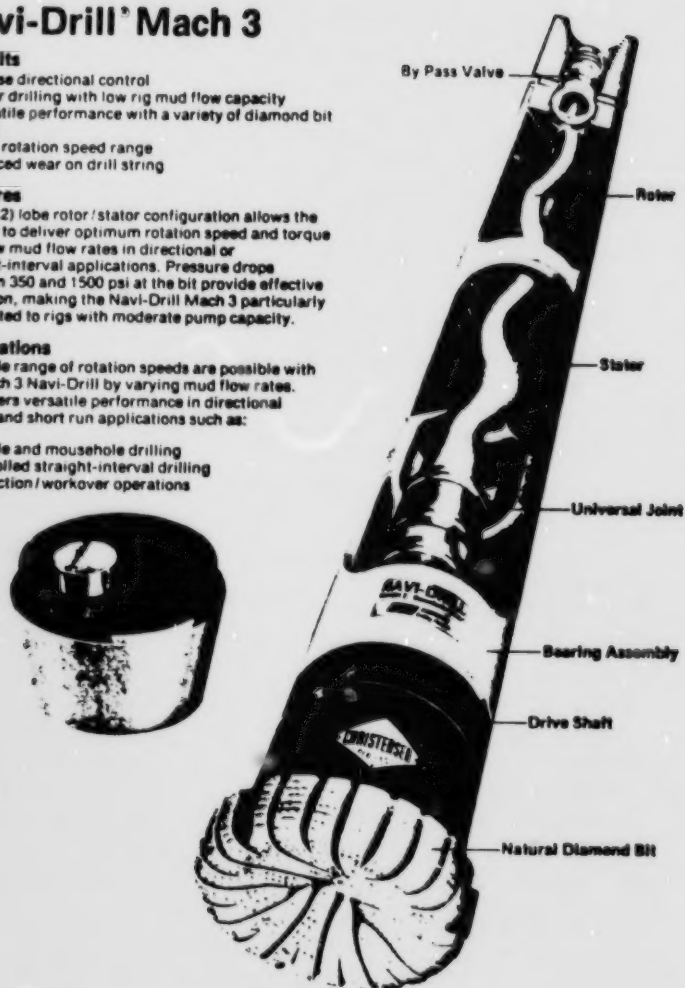


Figure 4-9

Downhole Rotor and Stator Motor

Source: Norton Christensen Navi-Drill

Norton Christensen Downhole Turbine

Benefits

- High rotation speed
- High Torque
- Fast penetration rates with diamond, compact and BallaSet™ bits
- Stabilized straight-interval control
- Long tool life
- Less wear on drill string and casing

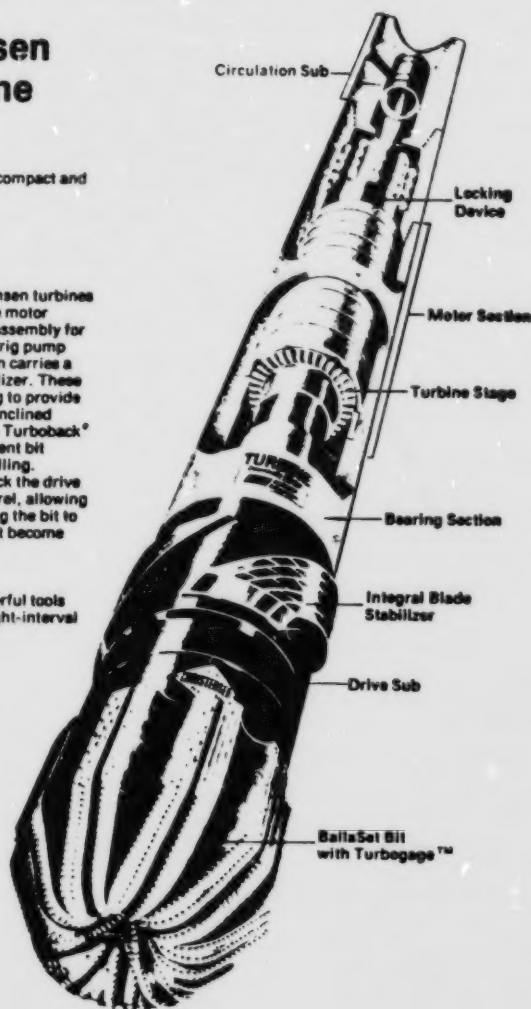
Features

Modular design gives Norton Christensen turbines added versatility by enabling up to three motor sections to be connected to the bearing assembly for increased drilling torque, depending on rig pump capacity. Each motor and bearing section carries a Norton Christensen Integral Blade stabilizer. These stabilizers are interchangeable on the rig to provide optimum versatility in either vertical or inclined drilling. In addition, Norton Christensen Turboback® stabilizers are available, and offer excellent bit stabilization for straight-hole turbine drilling.

A locking device can be activated to lock the drive sub and bit against the turbine outer barrel, allowing the entire bottom hole assembly including the bit to be rotated from the surface should the bit become stuck.

Applications

Norton Christensen Turbines are powerful tools designed for all high-performance, straight-interval drilling applications.



*Trademark of Norton Christensen, Inc.; patent pending.

Figure 4-10

Downhole Turbine Motor

Source: Norton Christensen Downhole Turbine

AR-124

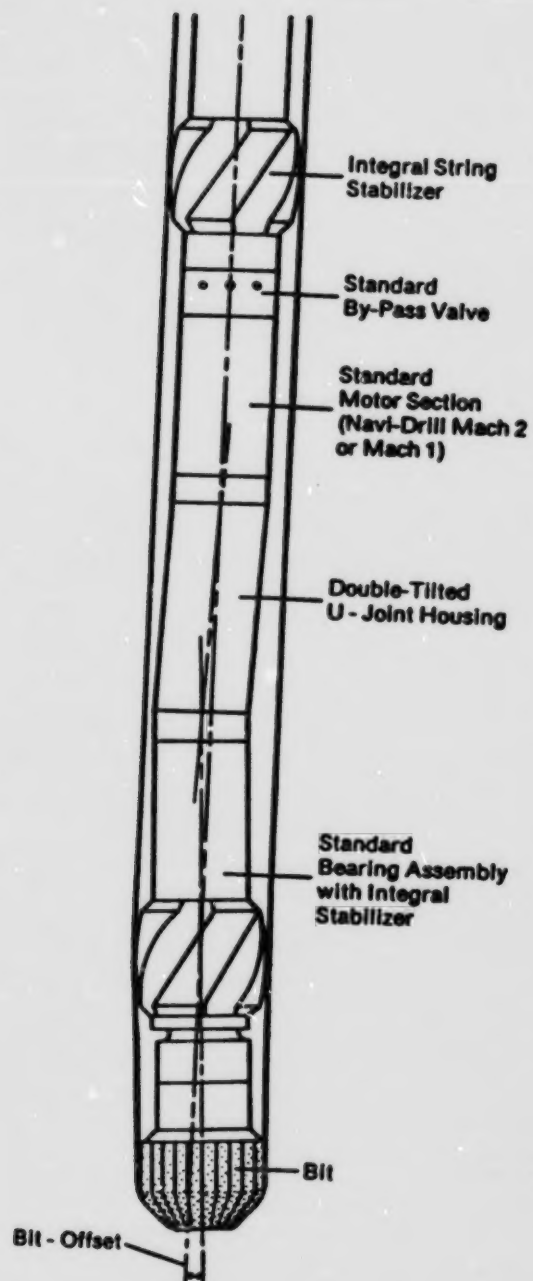


Figure 4-11

Navigation Sub with DTU Housing

Source: Norton Acc-Trac Directional Drilling Services

PRE-DRAINAGE IN COAL MINES

SECTION 5 - IN-SEAM GAS FLOW STIMULATION
(HYDRAULIC FRACTURING)

TABLE OF CONTENTS

5.1	INTRODUCTION	5-1
5.2	HYDRAULIC FRACTURE STIMULATION IN COAL	5-2
.1	Results From Underground Observations	5-2
.1	Fracture Orientation	5-3
.2	Fracture Widths	5-4
.3	Fracture Lengths	5-5
.4	Roof Stability	5-5
.2	Stimulation Pressure	5-6
.1	Fracture Tip Plugging	5-6
.2	Viscous Coal Slurry	5-6
.3	Multiple Fracture Propagation	5-7
.3	Post Treatment Production	5-7
5.3	TYPES OF STIMULATION	5-8
.1	Fluids	5-8
.2	Proppant	5-9
5.4	DISCUSSION	5-10

LIST OF FIGURES

5-1	TW-5 Borehole, Fracture Orientation, Oak Grove Mine, Alabama
5-2	EM-5 Borehole, Fracture Orientation, Emerald Mine, Penn.
5-3	Maximum Treatment Pressure Gradients for the Black Warrior Basin
5-4	Bottomhole Instantaneous Shut-In Pressure Gradient for the Black Warrior Basin
5-5	Relationship Between Maximum Treatment Pressure and Instantaneous Shut-In Pressure, for the Black Warrior Basin

PRE-DRAINAGE IN COAL MINES

5.0 IN-SEAM GAS FLOW STIMULATION (HYDRAULIC FRACTURING)

5.1 INTRODUCTION

Hydraulic fracturing is a process used to increase the deliverability of reservoir fluids to the drillhole. The object is to create a highly conductive fracture extending into the reservoir away from the drillhole. This is done by cracking the rock with pressure transmitted through a hydraulic fluid. The fracture is created when the fluid pressure becomes large enough to overcome both the minimum in situ stress and the tensile strength of the rock. The fracture is extended by pumping additional fluid at the high pressure. Because the pressure within the fracture is greater than that in the reservoir pore space, fluid leaks out of the fracture and into the reservoir. When pumping stops, all of the fluid leaks into the reservoir and the fracture closes. The fracture can be held open if a proppant (usually sand) is pumped along with the fluid.

The conductivity of the propped fracture is many times greater than that of the reservoir. This results in increased production rates, since the length of reservoir that the fluid must flow through to get to the fracture is less than the distance it would have to flow through to get to the drillhole.

Hydraulic fracturing has been used successfully by the oil and gas industry on sandstone and carbonate formation for many years. This stimulation process has more recently been applied by the mining industry to predrain methane from coal seams in advance of mining. Over 130 treatments (Jones et al, 1986) have been pumped; all treatments, however, have been conducted through holes drilled from surface. The fractures intercepted by mining reveal a great deal about the process of fracturing coal.

A variety of fracture fluids, volumes, pump rates and proppant concentrations have been tried. Fluids include 75% quality foam, gelled water and gelled water with CO₂. Rates have generally been between 16 to 32 L/s although some have been as high as 30 L/s. Treatment volumes range from 13 m³ to ones in excess of 200 m³. Up to 20 tonnes of sand have been placed at concentrations to 180 kg/m³. Post treatment results are encouraging with methane production from some wells as high as 76 L/s.

The information indicates that a good potential exists for predrainage of methane in Canadian coal mines. The Sydney Coalfield being submarine, presents a challenge in that the stimulation process must be conducted from within the mine. This will be a first in the industry since no reference to fracturing from within coal mines has been found.

5.2 HYDRAULIC FRACTURE STIMULATION IN COAL

The same process used to hydraulically fracture sandstones is used to fracture coal seams. Much is known about fracturing sandstones because it has been studied in detail by the oil industry. Coals, however, react somewhat differently because they are soft, friable and naturally fractured. For this reason, Jones et al (1986), mentions that there is no consensus among the industry as to the best approach for coalbed stimulation design. An advantage in learning about fracturing coal is that the created fractures can be mined through and the finished product can be seen. From the many treatments conducted, a data base has been collected to help provide some insight as to what can be expected from this type of stimulation.

.1 Results From Underground Observations

Many of the fracture treatments pumped into coal seams have been intercepted by mining. In most cases, a fluorescent paint is mixed with the fracturing fluid so that the fractures can be easily detected.

The number of fractures created from a single treatment along with their lengths, widths and orientation have been observed.

.1 Fracture Orientation

Diamond and Oyler (1987) outlined underground observations on 22 government sponsored treatments in the United States. All the created fractures had vertical components with about half having additional horizontal ones.

The horizontal fractures were generally found within bedding planes, most commonly on top of the seam. They have been found as deep as 349 m. In one case, four horizontal fractures were observed; however, one is more common.

The vertical fractures tend to follow the coal's cleat system. Usually, more than one fracture will occur. A primary set of fractures, which are longer and more numerous, will parallel the face cleat with a second set along the butt cleat.

A possible explanation for multiple fracturing is that as the pressure builds near the drillhole during propagation, it becomes large enough to overcome the stress in another direction. A new fracture, usually along the butt cleat, will then form. It will extend to provide cross link for the fluid to create more fractures along the face cleat. If the pressure becomes large enough to overcome the overburden stress, a horizontal fracture will form. Depending on the in situ stress in the overlying strata, the fracture could break out of the coal.

A particularly good example of multiple fractures is from a treatment conducted through the TW-5 borehole in the Blue Creek coalbed at the Oak Grove mine in Alabama (Diamond and Oyler, 1987).

Figure 5-1 shows the multiple fractures along the face cleat, a lesser amount along the butt cleat and a horizontal fracture at the top of the seam. Depth of this seam is 349 m. The treatment fluid was foam without the addition of proppant. Normally when proppant is used, it is found in only one main face or butt cleat fracture.

An example where multiple fractures were not found is from a treatment conducted on the EM-5 borehole in the Mary Lee Coalbed at the Emerald Mine in Alabama (Lambert et al, 1980). In this case, a single vertical fracture intercepted a horizontal fracture along the coal/shale interface at the roof of the seam (Figure 5-2). The depth of this seam is 335 m. The treatment fluid was foam with the addition of proppant.

Jones et al (1986) mention that only single fractures have been observed in the Jim Walters Mine in the Brookwood Coal Degas Field.

.2 Fracture Widths

The propped fracture widths have been measured between 2 and 50 mm with the average being 25 mm (Jones et al, 1986). In one case, however, at Borehole TW-2 in the Oak Grove mine, the maximum measured width was 114 mm. These widths are larger than those estimated in sandstone. A possible reason is that coal has a Modulus of Elasticity (E) that is 10 times less than sandstone. Theoretical fracture width is approximately inversely proportional to E (Nolte, 1986).

In situations where the fracture was traced with fluorescent paint beyond the point of proppant, the fracture width was essentially only the width of the naturally occurring cleat (Diamond and Oyler, 1987).

.3 Fracture Lengths

The overall length of any fracture will depend on the volume of fluid pumped. It has generally been found that the created lengths have been well short of the designed lengths. This is probably due to the common occurrence of multiple fractures. Current fracturing simulators do not accommodate this variation.

The maximum reported extension for a sand filled vertical fracture is 127 m. This occurred at Borehole I-NE in the No. 6 coalbed in Illinois. 2.9 tonnes of 10/20 mesh sand were pumped in 41.7 m³ of gelled water. The coal seam was 2.1 m thick.

The longest non-propped fracture reported to date is 192 m and occurred from a treatment at borehole DHM-6 in the Blue Creek coalbed in Alabama. The treatment size was 51 m³ of 75% quality foam. The seam thickness was 1.8 m.

.4 Roof Stability

In the cases reported, no existing damage to the roof can be attributed to hydraulic fracturing. In approximately half of the 22 treatments outlined by Diamond and Oyler, (1987) fluid entry into the roof has been observed. It appears, however, that it has always entered pre-existing joints.

In only one case had additional roof support been used. This occurred at borehole TW-3 at the Oak Grove mine in the Blue Creek coalbed. Lambert and Trevits, (1979) suggest that the fluid had entered a pre-existing roof joint since 57 other naturally occurring joints were found in the area. Also, cement used to set the casing was found in the same roof joint up to 25 metres away. Although additional roof support was installed, no roof fall was experienced during mining operations.

.2 Stimulation Pressure

Pump pressure during the fracturing treatment is important, not only for equipment design, but also to help determine the fracture orientation. Jones et al (1986) presented treatment pressure data for the Black Warrior Basin. Although there is a lot of scatter in the data, especially at the shallower depths, the maximum injection pressure gradients are very high. The data ranges from 22 to 50 kPa/m (Figure 5-3). These gradients are considerably higher than the instantaneous shut-in pressure (ISIP) gradients which is the pressure gradient immediately after pumping (Figure 5-4). The ISIP gradients for depths greater than 450 m range from 12 to 18 kPa/m and for depths between 250 m to 450 m, the ISIP gradients climb to a maximum of 27 kPa/m. Figure 5-5 shows the comparison of the maximum treatment pressure with shut in pressure. Jones et al (1986) hypothesize as to the causes of the higher pumping pressures.

.1 Fracture Tip Plugging

Coal fines which are generated from the fracture face during pumping can plug the tip of the fracture. These fines will prevent the full fluid pressure from acting at the tip and therefore, cause the pressure required for extension to increase.

.2 Viscous Coal Slurry

Coal particulate generated from the walls during pumping can increase the viscosity of the fracture fluid. Increased fluid viscosity results in greater pressure drop along the fracture, requiring higher pressures at the drillhole for propagation.

.3 Multiple Fracture Propagation

If pressure at the drillhole increases to the point where the intermediate or maximum principal stresses are overcome, the fracture will divert. The result, which has been confirmed from underground observation, is a matrix of fractures which tend to follow the face cleat. The face cleat is often almost perpendicular to the minimum stress. The pressure drop of a fluid flowing through the matrix of fractures will be greater than that through a single fracture of equivalent width. Also, the increased pressure from the side fractures cause a poroelastic back pressure effect. The effect is the same as would be experienced if the pressure within the pore space was very high.

.3 Post Treatment Production

Lambert et al (1980) provides a table of results on 71 separate treatments. Gas rates from a single borehole after stimulation have been as high as 76.4 L/s (6600 SCM/day) with the average being somewhere between 17 to 35 L/s (1500 to 3000 SCM/day). In most cases, before fracturing, the well was either not produced or gas flows were very small. Water production always exists, but is largest at the initial stages because coal is generally water saturated. Conversely, the gas production increases as the water content drops because the relative permeability to gas in coal is small when the water saturation is high.

It is important to keep the pressure at the coal seam as low as possible so that the gas will desorb from the coal. Downhole pumps are required to produce the water and keep the hydrostatic pressure down since all fracturing treatments have been conducted through holes drilled from surface.

5.3 TYPES OF STIMULATION

All fracture treatments require some type of hydraulic fluid and usually a proppant to hold the resulting fracture open.

.1 Fluids

The best fluid is one that has sufficient viscosity to carry the proppant and inhibit leak-off to the surrounding reservoir. Gelled water is commonly used because its viscosity can be high during pumping and then reduced during flowback. With the addition of a crosslinker, viscosities can be increased to values in excess of 250 mPa.s.

In some cases, an energizing fluid is required to enhance flowback. This fluid can be created by adding a gasifying agent to another fluid such as gelled water. When pumping is complete and flowback is initiated by releasing the pressure, the gas expands and pushes the fluid out of the hole. Both CO₂ and N₂ have been used for this purpose. CO₂ is pumped as a liquid, but turns to a gas during flowback, N₂ is mixed with water as a gas.

When N₂ is blended in the correct proportions with gelled water containing surfactants, a foam is created. The benefits of this foam are high viscosity, which provide good sand carrying capability, reduced fluid leak-off and quick fluid recovery from the expansion of the N₂. Approximately 63% of the treatments outlined by Lambert et al (1980) and 80% of those outlined by Diamond and Oyler (1987) have used foam. Foam viscosity is not measurable by normal viscometers; however, it is estimated to be in excess of 100 mPa.s. Blauer and Kolhaas (1974) discuss foam viscosity.

.2 Proppant

In sandstone formations, a proppant is required to hold the fracture open after pumping. The majority of the treatments pumped into coal seams have also used proppant. There is some question, however, as to whether proppant is really necessary.

In contrast to sandstone, the cleat structure in coals cause them to be friable. As fracturing fluid is pumped into the formation and then flowed back, coal fragments break off and move along the fracture. an example of an extreme case was observed during flowback operations following a foam stimulation of the Glover #1 well in the northern San Juan basin (Jones et al, 1986). Approximately ten tonnes of predominantly coal fines (up to 2 cm diameter) flowed to surface with the returning fluid.

It is probable that some of the generated fines stay in the fracture and act as a proppant. For this to be effective, however, the closure stress must be small because the coal particulate has very little compressive strength. The closure stress is the total stress less the fluid pressure within the seam. As the well is produced and the pressure is drawn down, the closure stress increases and tends to close the fracture. This reduces the permeability of the fracture. Gray (1987), however, shows that in some coals, as pressure is reduced and gas is desorbed, shrinkage occurs; this tends to reduce the closure stress and increase permeability. Production characteristics in the Oak Grove mine suggest that closure stresses are low since sand flow into the wellbore is common throughout the life of the well.

Mahoney et al (1981) outline a situation at the Oak Grove mine in which proppant was not used. Combined production from two boreholes averaged over 23.1 L/s (2000 SCM/day) over an eleven month period.

During the development of this mine section, the gas removed from the ventilation system amounted to 40% less than that removed from a similar section in the vicinity, but outside the drainage area of the boreholes.

5.4 DISCUSSION

From the many hydraulic fracturing treatments conducted on coal seams, it has been found that this method of stimulation is beneficial in pre-draining methane in advance of mining. Methane rates from a single borehole have increased to an average of 17.4 to 34.7 L/s (1500 to 3000 SCM/day). One borehole in the Mary Lee coalbed produced in excess of 76.4 L/s (6600 SCM/day).

As yet, no roof damage can be attributed to the treatments. In one instance, additional roof support was used; however, it is believed the fracturing fluid entered a pre-existing roof joint since 57 additional roof joints were found in the area. No problem with roof falls occurred during mining operations.

Variations in injection pressures from well to well and underground observations of actual fractures show that many inconsistencies occur when fracturing coal. This is a result of the large variations in coal properties. As opposed to sandstone, coals are soft, friable and naturally fractured. There are no simulators which handle this situation, making it difficult to conduct accurate treatment designs.

The first hydraulic fracturing treatment to be pumped from within mine workings should be kept simple.

- Gelled water should be used as opposed to foam. Although larger volumes of water will be required, the need for N₂ storage facilities and pumping equipment will be eliminated.

- An effort should be made to quantify the stress field in both the coal and overlying strata. From this, a prediction can be made as to whether the fracture orientation changes or penetrates the roof. Pressure restrictions can also be set to ensure roof penetration does not occur.
- Accurate monitoring of injection pressure must be done. With drillholes nearly horizontal and relatively short, the injection pressure at the pumps will be very close to the fracturing pressure at the seam. Hydrostatic pressure changes will not occur and there will be very little frictional pressure loss.
- Proppant should not be used. Good results have been obtained from treatments conducted without proppant and the need for sand storage and blending facilities will not be required. Also, by not using proppant, the potential for a screen-out is eliminated and injection rates can be controlled to keep pressures down.

AR-124

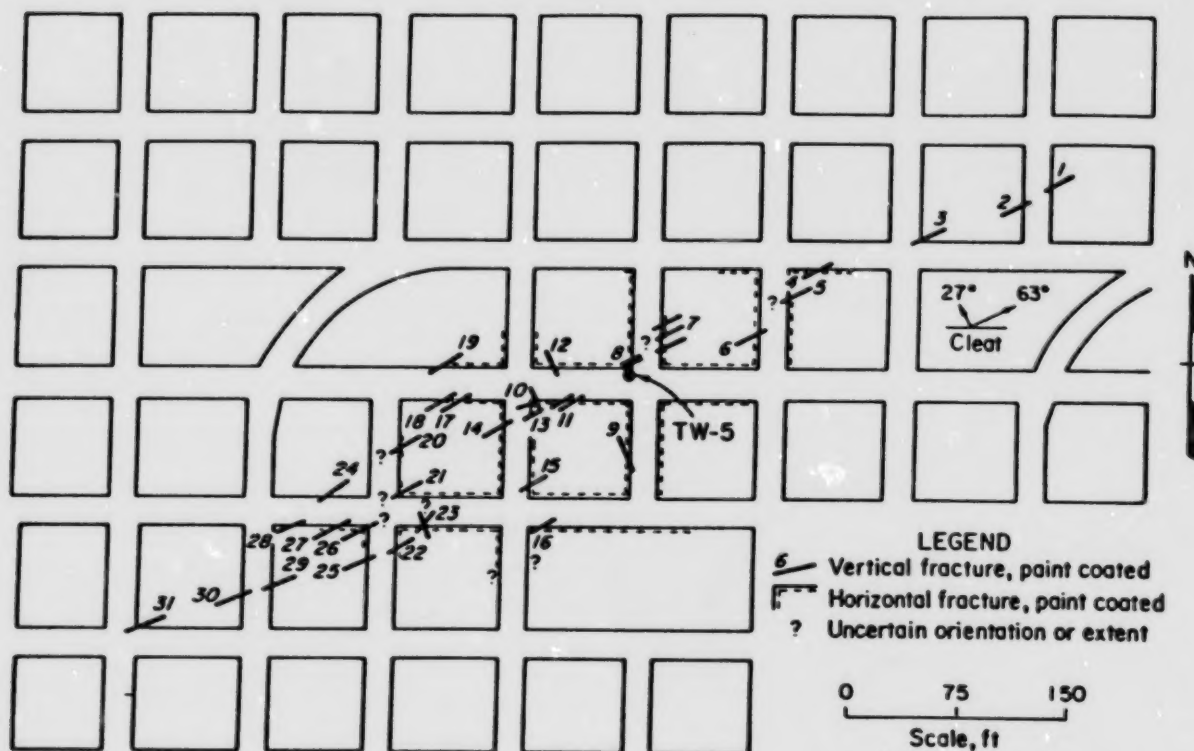


Figure 5-1
TW-5 Borehole, Fracture Orientation, Oak Grove Mine, Alabama
 After: Diamond and Oyler, 1987

AR-124

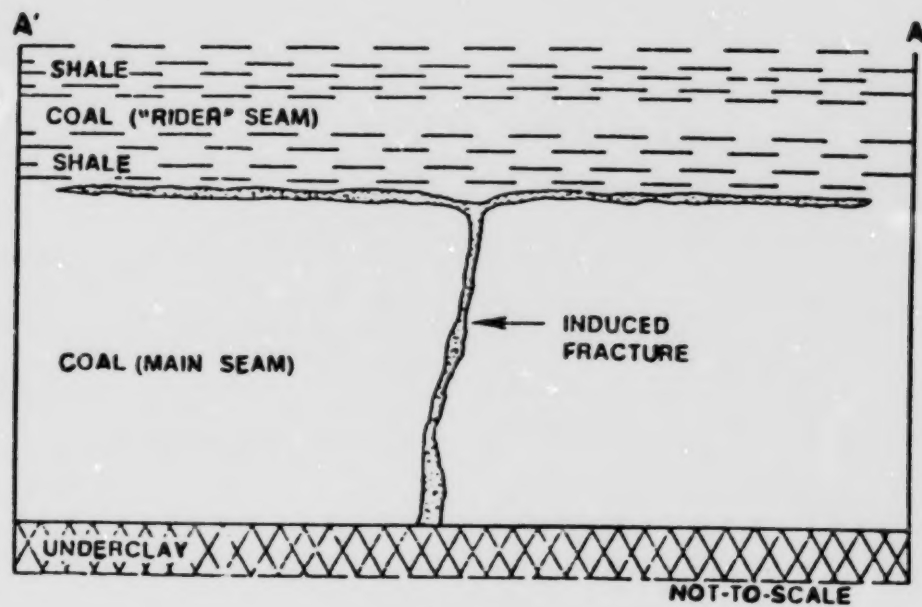
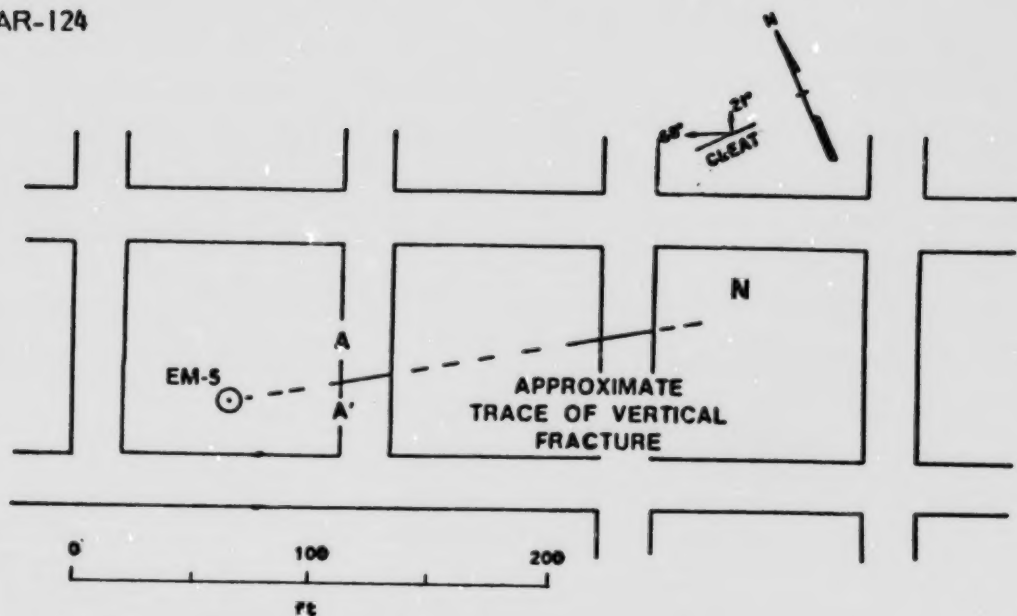


Figure 5-2
EM-5 Borehole, Fracture Orientation, Emerald Mine, Penn.
After: Lambert, 1980

AR-124

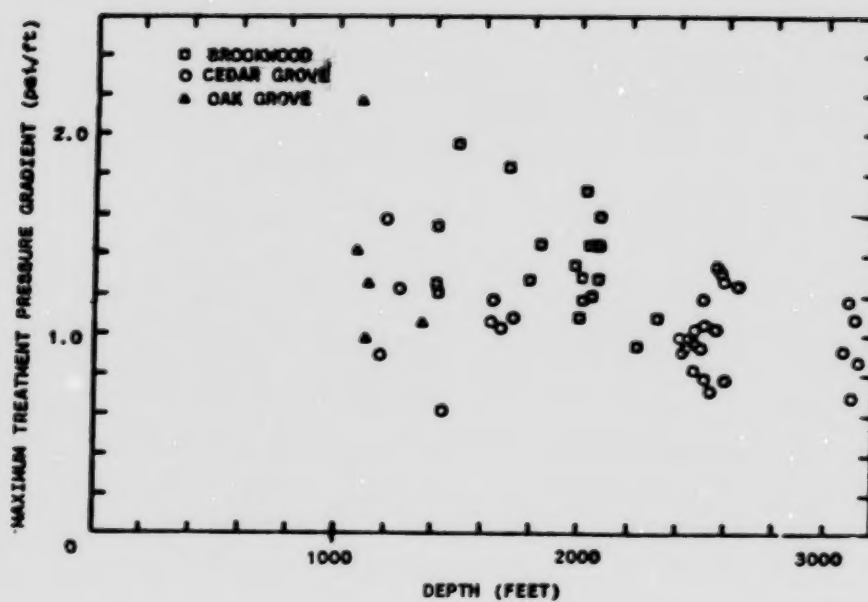


Figure 5-3
Maximum Treatment Pressure Gradients for the Black Warrior Basin
From: Jones et al, 1986

AR-124

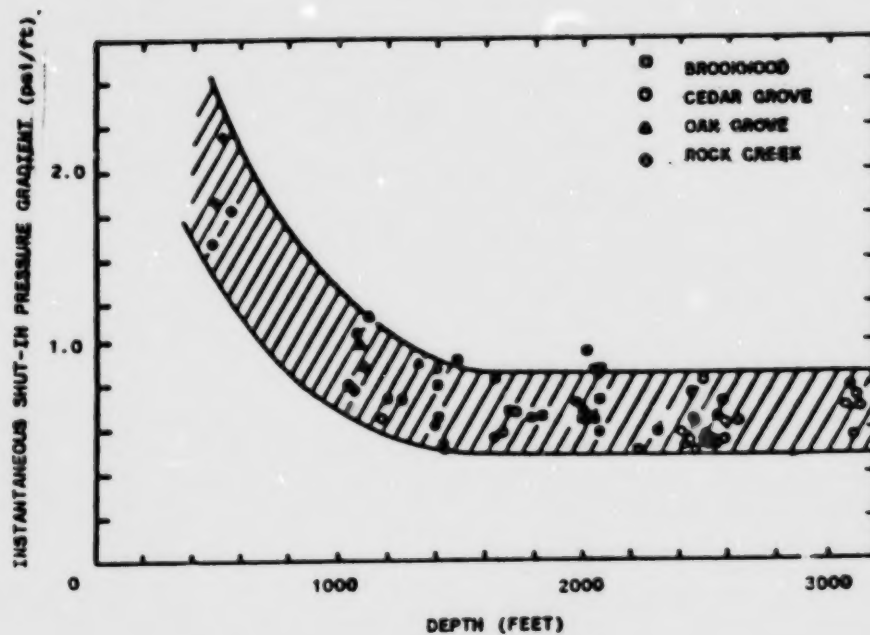


Figure 5-4
Bottomhole Instantaneous Shut-In Pressure
Gradient for the Black Warrior Basin
From: Jones et al, 1986

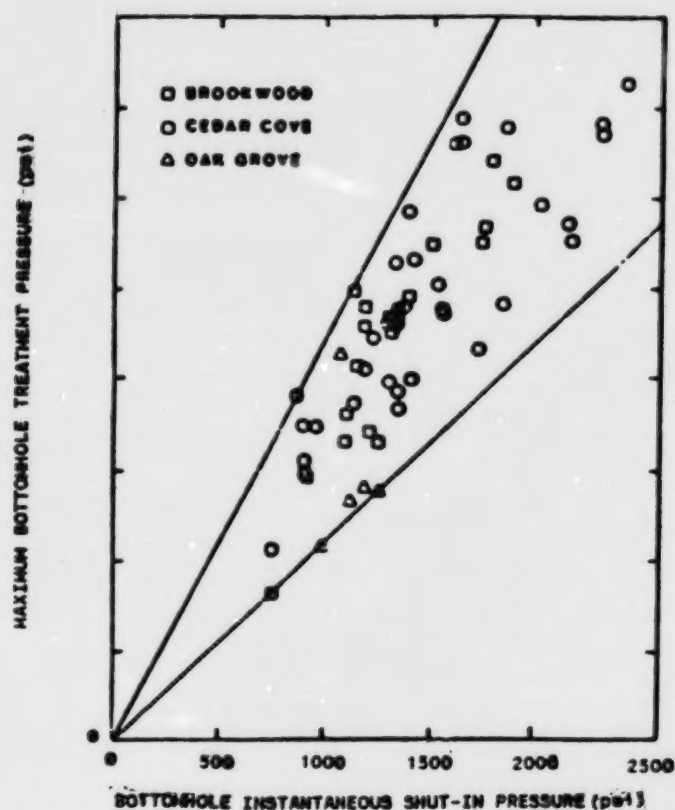


Figure 5-5
 Relationship Between Maximum Treatment Pressure and
 Instantaneous Shut-In Pressure, for the Black Warrior Basin
 From: Jones et al, 1986

AR-124

PRE-DRAINAGE IN COAL MINES

SECTION 6 - DISCUSSION AND CONCLUSIONS

TABLE OF CONTENTS

6.1	THE COAL SEAM	6-1
	.1 Coal/Methane System	6-1
	.2 Permeability	6-2
6.2	DRILLING	6-3
6.3	HYDRAULIC FRACTURING	6-4

PRE-DRAINAGE IN COAL MINES

6.0 DISCUSSION AND CONCLUSIONS

6.1 THE COAL SEAM

.1 Coal/Methane System

Coal as a reservoir material has a very low porosity (5%), but a very high concentration of methane mostly held in an adsorbed state (Jones et al, 1986). The methane content, for example, could be 18 SCM/m³ of coal whereas a sandstone reservoir with the same 5% porosity would contain about 6 SCM/m³.

The methane gas is associated with the coal as adsorbed molecules on coal (90%) and as free gas in the pore space (10%). These two phases reach an equilibrium condition within a static physical environment. In order to release adsorbed methane, the conditions must be unbalanced, the simplest way to achieve this for in-situ coal is to reduce the pressure on the reservoir. Kahil and Masszi (1982) point out that "regardless of how high the permeability, methane production from coal cannot exceed the desorption rate".

The Harbour Seam in the Sydney Coalfield has a relatively low desorption rate. The seam contains about 12 SCM/m³ of mostly methane; current estimates are that about 3 - 4 SCM of CH₄ per m³ of coal are released at the time of coal cutting on the longwall face. The rest is desorbed slowly. Harbour Seam coal after passing through the coal preparation plant is required to be stockpiled for weeks before it can be loaded into shipholds.

No data was available for gas in the Phalen Seam. Current planning, however, anticipates a similar gas content to that of the Harbour Seam.

The Hub Seam, mined in the Prince Colliery, does not appear to be a gassy seam, based on practical mining conditions. However, no data is available.

.2 Permeability

Reduced pressure on the reservoir can only release adsorbed gas if the permeability is sufficiently high for the gas to migrate through the reservoir. Two parameters strongly influence the reservoir: confining pressure and water content.

Coal as a physical material has a very low elastic modulus ($E = 3.2 \times 10^{-6}$ kPa) and a high Poisson's Ratio (0.33) (Jones et al, 1986). This means that relatively small changes in confining pressure can have significant effects on void space in the coal reservoir. Coal cleat, for example, imparts a fracture permeability to the reservoir; however, abutment loading due to mining can create permeability barriers. Predrainage trials have shown that a seam over- or under-worked releases methane much more readily than the same seam under virgin conditions, with the same depth of cover. The reduced confining pressure is thought to allow the cleat system to open.

Phalen Colliery in Cape Breton is the only mine currently working in an area over-mined by Langan and No. 14 Collieries. Minimum vertical load would be anticipated in 1W, 2W and 3W Panels under the depillared area of No. 14 Colliery. Another area in the future would be the 4E and subsequent panels which are aligned below Langan 1E and lower panels. Any in-seam predrainage holes must be long enough to penetrate mining induced abutment loads.

Water severely inhibits gas flow in a coal reservoir (Section 2.4). All predrainage trials have noted the necessity to drain water from the reservoir in order to achieve maximum gas production. In-seam predrainage holes should preferably be drilled to the rise. This would

allow for simple water drainage through a water trap on the drainage line.

6.2 DRILLING

Methane predrainage can be accomplished via openings in an unmined block of coal (development roadways, auger holes or drillholes). Long drillholes (up to 1000 m) can now be drilled with some success in reaching a defined target. However, seam geometry is often not precisely known and attempts are then made to maintain a drillhole in a undulating horizon - an unenviable task. The current state of the art precludes drilling holes in-seam for the length of a longwall panel (2000 m) staying between gate roads 200 m apart and within the seam.

The alternative is to drill short holes across the panel. A retreat longwall system is ideal for this purpose since drilling can be carried out from the development gate for the next panel. The appropriate spacing between holes, the grouted casing length and the lead time for effective drainage will have to be determined by sample and in-situ testing.

The laboratory testing requirements limit the initial drilling to a rotary system capable of recovering core. If additional in-situ experiments, such as hydrofracturing, are planned, the drillholes must be of sufficient size to allow for this activity. An HQ core barrel and drill string (hole diameter 96 mm, core diameter 63 mm) would accommodate these requirements.

6.3 HYDRAULIC FRACTURING

Hydrofracturing coal reservoirs to date has been carried out exclusively from surface. The treatments have benefited from all the sophistication developed in the petroleum industry including: nitrogen foam, sand proppant and pumping capacities for large volumes of fluid. Fractures that have been intersected by subsequent mining operation have shown that:

- The maximum length was 192 m using 51 m³ of foam in a 1.8 m seam.
- Vertical fractures tend to follow the cleat direction as multiple fractures.
- Roof strata is in a different stress regime than the coal seam, i.e., fractures do not pass from coal to roof strata unless roof joints are in evidence.
- No evidence has been found to date that roof conditions deteriorate due to hydrofracturing.

Hydrofracturing in Cape Breton Collieries will be the first fracturing treatment for gas drainage done from underground. The philosophy adopted in this study has been to keep the initial trial simple. This approach results in the following basic constraints:

- Use gelled water as fracturing fluid - this eliminates nitrogen storage and pumping in the mine.
- Use no proppant initially - the literature is not unanimous in the need for proppant when fracturing coal since multiple fractures are formed and coal particles are found in the fracturing fluid. This decision reduces the equipment required to blend sand at a preset rate into the treatment fluid.

Hydraulic fractures open up perpendicular to the local minimum stress direction:

- At shallow depth, these fractures will be near horizontal.
- At depth they will be vertical probably along the cleat.
- Fractures will not forcibly break out into mine openings since the minimum stress is perpendicular to the rib and a fracture would, therefore, parallel the rib line.

PRE-DRAINAGE IN COAL MINES
SECTION 7 - CAPE BRETON TEST PROGRAM
TABLE OF CONTENTS

7.1	LINGAN AND PHALEN COLLIERIES	7-1
7.2	PROGRAM OBJECTIVES	7-1
7.3	PHALEN TEST PROGRAM	7-2
.1	Phase 1 - Drilling and Drainage	7-2
.1	In the Mine	7-3
.2	In the Laboratory	7-3
.3	The Research Program	7-3
.4	Initial Design	7-5
.2	Phase 2 - Drilling and Monitoring	7-9
.3	Phase 3 - Hydrofracturing and Gas Drainage	7-9
.1	Design Approach	7-9
.2	Treatment Design	7-9
.3	Hydrofracturing Equipment	7-12
7.4	COST ESTIMATE	7-13

TABLES

7-1	Reservoir and Treatment Data
7-2	Phalen Phase 1 - Cost Estimate
7-3	Phalen Phase 3 - Estimate Cost of Initial Hydrofracturing Test
7-4	Estimated Cost of Hydrofracturing Equipment

FIGURES

7-1	Sydney Coalfield
7-2	Isometric View of Four-Hole Drainage Trial
7-3	Drillhole Layout - Test Program
7-4	Pressure Buildup Test Equipment
7-5	Phalen Phase 1 - Test Program Schedule
7-6	Phalen Program Schedule
7-7	Pump Skid Proposal
7-8	Batch Tank 10 m ³
7-9	Batch Tank 10 m ³
7-10	Hydraulic Fracturing Equipment Layout

PRE-DRAINAGE IN COAL MINES

7.0 CAPE BRETON TEST PROGRAM

7.1 LINGAN AND PHALEN COLLIERIES

The Lingan-Phalen Resource Block (Figure 7-1) currently supports two separate mining operations of the Cape Breton Development Corporation (CBDC): the Lingan Colliery in the Harbour Seam mining at -650 m elevation four kilometres from shore and the new Phalen Colliery in the Phalen Seam mining at -300 m elevation one kilometre from shore. The Phalen Seam lies 140 m stratigraphically below the Harbour Seam. Both the Phalen and Harbour Seams are considered to be gassy.

Lingan Colliery operates three advancing longwall faces; few development roadways are available under this system to access future mining panels. CBDC plans to change to retreat longwall panels in the foreseeable future. Lingan Colliery workings should then be the site of a second test program.

Phalen Colliery operates two retreating faces. In order to test methane drainage under a reduced overburden pressure, the first test site should be in 2W or 3W Panel. This area has been overmined by No. 14 Colliery room-and-pillar operations with pillar extraction in the Harbour Seam.

7.2 PROGRAM OBJECTIVES

This test program addresses only the pre-drainage of gas from the working seam, CANMET/CBDC will be addressing alternative measures to reduce gas emissions under other programs. The objectives include:

- capability to fully core drillholes in seam.
- capability to maintain 50 to 100 m drill holes in the seam.
- determine the type of coal sample testing required to assess the drainability of the seam.

- in-situ testing of gas pressure and flows within or between drillholes.
- gas drainage and monitoring of in-seam effects.
- hydrofracturing and in-seam monitoring of the 'frac' ("fracturing" from within the working seam has not been attempted to date).
- gas drainage and monitoring of in-seam effects in the frac'd drillhole.
- assess the effectiveness of methane drainage.

The cumulative data should be used to design a pre-drainage program and assess its economics.

A complete test program for the Sydney Coalfield must address both the deep Harbour Seam coal in Ligan mine and the shallower Phalen Seam, stress relieved by overmining in some locations. The ease of access and availability of development roadways on the down dip side of longwall panels makes Phalen Colliery the preferred site for the initial test program.

7.3 PHALEN TEST PROGRAM

The Phalen Test Program should be addressed in three distinct phases:

1. Drilling (3 holes) core testing, methane monitoring.
2. Drilling (3 holes in the same panel as above), specific core testing, methane monitoring.
3. Hydrofracturing in drillholes (2. above), methane monitoring.

.1 Phase I - Drilling and Drainage

This program is a research test program to gain some knowledge about the Phalen Seam gas reservoir; it should not be considered or evaluated as a pre-drainage trial.

The test program schedule for Phase I is given in Figure 7-5.

.1 In the Mine:

- Select test site.
- Evaluate drilling procedures to drill holes in coal for gas drainage.
- Evaluate the lengths of holes to be drilled.
- Determine the economics of drilling holes for gas release.
- Measure methane flows from the holes without hydrofracturing.

.2 In the Laboratory:

- Determine the gas content of the coal samples.
- Determine Darcy flow and diffusional movement.
- Determine the relationship between stress and permeability.
- Determine fluid pressures and equivalent sorption pressures.

.3 The Research Program:

The laboratory testing and the drilling/monitoring in the mine are interactive; underground drilling will provide coal samples for laboratory testing.

A program similar to the research proposed was carried out in mines in northern and central Queensland, Australia, and is described by Gray (1987) as part of studies on reservoir engineering in coal seams.

The Phalen - Phase I program specifies:

- Site selection.
- Drillhole size and spacing.
- Laboratory testing requirements.
- Equipment selection.

- Services required.

Drillhole diameter - The size of drillholes is determined by tests necessary to obtain required information. i.e., For samples required for laboratory testing:

<u>Laboratory Test</u>	<u>Core Size</u>
Permeability Tests	100 mm
Coal Substance Desorption	Cuttings and Core up to 100 mm
Measurements on the Effect of Stress	100 mm
<u>For In-Mine Testing</u>	<u>Hole Size</u>
Abutment zone determination	100 mm to 50 mm
Water and gas makes	100 mm to 50 mm
Effect of hole size	100 mm and 50 mm
Effect of stimulation by Hydrofracturing	Min. 100 mm

The HQ size would be most appropriate with hole diameter of 96 mm and core diameter 63.5 mm.

The precise size would be dependant on actual drilling equipment used but should be approximately as suggested.

Drillhole Spacing

Within the literature tests have been made on blocks of coal with general hole spacings varying between 5 m and 20 m. For the Australian coals, the main program was 12 m spacing with one central hole at 6 m for pressure testing (Figure 7-2).

The length of holes in the Australian program was 80 m. The object was to ensure that the abutment zone was penetrated. The position of the abutment zone can be determined in the first drill hole. Indications of the abutment zone may be by penetration

rates of drilling at monitored pressures, or it may be necessary to make incremental flow tests on the initial drillhole. The actual program would be adjusted as information is produced.

.4 Initial Design

The initial design would be for three drillholes. The first two holes to be at 5 m centres with the third hole tentatively 10 m away (See Figure 7-3).

As the effect of water on gas drainage is to be determined the holes should be drilled slightly to the rise with provision for drainage on each hole. The first holes should be drilled across the main cleats and within the seam.

The program should develop incrementally under control supervision, using experience gained in each operation to refine procedures for the following hole.

- Hole No. 1

Drillhole diameter should be HQ size. Both the hole and the seam should rise, and the hole should be in the order of 0.75 m from the roof of the seam. Details of collaring the holes are given in Section 4. The collar should be 6.1 m in length inserted in a hole 150 mm diameter drilled for 6.0 m.

A core sample should be taken near the collar and the sample should immediately be sealed in polythene and taken to the surface for laboratory testing. The precise location of the sample should be noted.

When the collar and stuffing box with water/gas traps and pressure gauges is installed, the hole should be incrementally drilled and tested. Penetration rates at noted drilling pressures should be carefully logged. At approximately 10 m intervals, further samples of core should be sealed immediately and taken to the laboratory for testing. If possible, the sample lengths should be 0.3 m or more. During the course of drilling, fresh drill cuttings should also be sealed and taken to the laboratory. Again, the position of the bit at the time of the cuttings should be noted. Drill cuttings should also be continuously examined during drilling to confirm the hole is within the coal seam. The hole must be drilled at least 50 m in coal and preferably up to 100 m.

This initial hole would then be available for comparing laboratory data with in-mine information.

While the hole is fresh, readings should be taken of pressure build-up and combined pressure and flow testing (Figure 7-4). A complementary test can be made using the air injection technique (Barron, 1978). This is a useful test to assess leakage from the hole. The Australian experience was that typical flow and buildup periods were 1 to 2 days).

Gray (1987) notes that up to five pressure distribution tests at 10 m intervals can be made in a 7 hour shift. The Australian experience was that packers had a short life and were difficult to maintain; for long term study, grouted-in pressure sensors were installed. (In the case of the Cape Breton program, this is an option for a later stage). We propose that flows be allowed to continue through the water traps and they be measured against time. Samples of the gas should be taken initially at daily intervals and analyzed, preferably by gas chromatograph, for constituent gases.

The drill core should be carefully examined for major cleat and joints and spacings recorded so that an attempt can be made to correlate them with future holes.

- Laboratory Testing
 - Desorption of core samples in canisters.
 - Desorption of cutting samples in canisters.
 - Permeability tests of core samples - gas and water.
 - Determination of total gas content of coal.
 - Analysis of gases.
 - Stress - permeability relationships.
 - Determination of equivalent sorption pressure - related to the coal matrix state of expansion and contraction.
 - Determination of strain parallel to the core axis by desorption and resorption
 - Repeat tests on samples from along the drillhole to establish effect of distance and abutment loading
- Hole No. 2

It should be drilled parallel to Hole No. 1, 5 m away

Procedures adopted at Hole No. 1 should be repeated or improved upon. In addition, Hole No. 1 should be allowed to flow free initially and continue to be monitored for gas flow as drilling takes place in Hole No. 2 (should the drilling of Hole No. 2 induce pressure on Hole No. 1 such that drilling fluid is lost, then Hole No. 1 would be shut-in until drilling and pressure testing has been completed in Hole No. 2).

Core samples and laboratory testing can be compared between the holes and at known distances from the drill collars.

Alternative options will then be available depending on conditions, e.g.

- Both holes may be allowed to flow free, being monitored for flow and pressure until steady state is observed
- Both holes can be shut-in and then released sequentially to observe any differences
- A known suction may be applied to Hole No. 1 and the effect on Hole No. 2 observed
- Suction may be applied to both holes and flows determined

In-mine observations can be compared with estimates of material balance from the holes.

The analysis of results from the two holes should establish a measure of permeability and drainage that can be expected from a hole spacing of 5 m. The results will influence the position of the third hole.

- Hole No. 3

If permeability is found to be high, the third hole could be sited 15 m away from Hole No. 2 but should there be little inter-hole influence the third hole should be 10 m away.

We propose that the procedures used for Holes No. 1 and 2 be repeated for Hole No. 3. The coring, sampling, pressure testing and flow monitoring should be carried out as between Holes No. 1 and 2.

.2 Phase 2 - Drilling and Monitoring

After a period of four to six months monitoring the initial three holes, three more holes will be drilled. The procedures experienced in drilling and the results of the monitoring will be used to determine the required spacing between the holes. At this time, it will also be possible to review the lengths and specifications of casing.

A further period of monitoring the six holes with testing of gas flows both from and between holes will be required before deciding on a hydrofracturing test.

.3 Phase 3 - Hydrofracturing and Gas Drainage

.1 Design Approach

The fracturing process in coals is somewhat different than in sandstones. Coals are naturally fractured, have dual permeabilities (cleat and matrix) and may be soft and friable. Experience has shown that injection pressures during treatments are somewhat higher, multiple fractures often occur, and created fractures are almost always shorter than predicted.

Design optimization on computer has limited application since there are no simulators which can describe the fracturing process in coal (Jones et al., 1987). The best approach for treatment design is, therefore, based on past experience keeping in mind that fracture treatments conducted in the Sydney Coalfield must be done within the mine.

.2 Treatment Design

Initially, the treatments should be kept simple and small. The objective is to determine if this type of stimulation will enhance

methane drainage. A list of the treatment and estimated reservoir parameters is given in Table 7-1.

- Fluid Type

The two basic fluids used in past treatments are foam and gelled water. To create foam, compressed air or nitrogen is required at rates as high as 275 SCM/min and pressures to 26,000 kPa. The total volume of compressed gas will depend on the pressure and size of the treatment. This requires additional equipment and cost; therefore, initial treatments should be conducted with gelled water only.

- Fluid Volume

Past treatment sizes have ranged from 13 to 230 m³. The small treatments (13 to 20 m³) were used when the boreholes were within close proximity of mining. A small amount of proppant (0.7 tonnes) was normally pumped with the fluid. Post treatment production increased to 4.6 to 6.9 L/s (400 to 600 SCM/day). Observed fracture lengths from the small treatments were usually less than 15 m for coal thickness of 1.5 to 2.0 m.

For a test program in the Sydney Coalfield, 20 m³ treatments should be pumped. Fracture lengths will remain small; however, they should reveal whether this form of stimulation will provide improved drainage.

Fluorescent paint should be added to the fluid so that the fracture can be mapped in the event it is mined through.

- Proppant

In most cases, proppant has been used; however, increased methane production has been obtained in situations where it was not. For simplicity and reduced costs, proppant should not be used initially. If it is found to be necessary, blending equipment can be added to the system at a later date.

- Injection Rate

For gelled water treatments pumped into coal seams from surface, injection rates have ranged from 0.7 to 1.6 m³/min. The injection rate for the test program will depend on the pressure; however, equipment should be capable of pumping 1.5 m³/min at 30 MPa.

- Pressure

The pressure will depend on the in-situ stress of the seam being fractured and on the overlying strata containing the fracture. It would be preferable to contain the fracture in the coal to ensure no damage is done to the roof. To do this, the minimum stress in the roof rock should not be exceeded.

Golder Associates conducted in-situ stress measurement on strata in the Prince, Langan and No. 26 Mines. Testing was done using both hydrofracturing and over coring techniques.

The strata tested in the Prince Mine was sandstone overlying the Hub Seam 222 m below sea level. The minimum principal stress was found to be near vertical agreeing well with the overburden stress (5-6 MPa). The major stresses, which are almost horizontal and near isotropic, are approximately twice the minimum stress. In the horizontal holes where hydrofrac-

turing tests were conducted, horizontal fractures were always created. In one instance where a mudstone layer was encountered, breakdown could not be achieved at 32.3 MPa. The maximum injection pressure obtained from one test conducted in the coal itself was 4.2 MPa with no distinct breakdown pressure.

At the Lingan Mine, tests were conducted in the siltstone above the Harbour Seam 593 m below sea level. Only horizontal fractures were induced. In several cases, breakdown did not occur at pressures levels varying from 20 to 30 MPa. The principal stresses were 33.8 MPa, 14.5 MPa and 12.2 MPa. The minimum stress dipped at 55°. No fracturing tests were tried in the coal.

At No. 26 Colliery, only overcoring tests were conducted in the strata above the Harbour Seam 762 m below sea level. The principal stresses were near isotropic ranging from 23.2 MPa to 26.3 MPa with the minimum being near horizontal. The calculated overburden stress was 21.7 MPa.

Past experience suggests that both horizontal and vertical fracture components will occur at the depth the fracture tests will be conducted in the Phalen seam. Little information has been obtained about the stress contrast in the coal and overlying strata. The fracture should however, remain in the coal since even if the stresses in both the coal and roof rock are the same the tensile strength of the roof rock will be higher.

.3 Hydrofracturing Equipment

Newsco estimate the fluid requirements to perform experimental hydrofracturing in the coal at Cape Breton to be a flow of 1.4 m³/min at a pressure of 18 MPa.

This would require a pump motor of 600 hp. There is a size/weight constraint of the CBDC underground transportation system of Height 1.4 m (4'8") x Width 1.52 m (5'0") and Length 9 m (30'0").

The maximum weight per unit is 15 t (33,000 lb). The equipment can be skid mounted - see Figures 7-7, 7-8, 7-9, and 7-10.

Two tanks would be required to provide a volume of about 10 m³ each. The tanks are each sized at 1.4 m x 1.5 m x 6.3 m.

To boost the fluid a 60 HP 230/460 VAC Mission 3x4 pump or a FWI C4500 pump mounted on one tank skid would be required.

7.4 COST ESTIMATE

An estimate of costs for a program to obtain knowledge of drainage characteristics is one coal seam together with the cost of providing and operating a test of hydrofracturing drainage holes are presented in Tables 7-2, 7-3 and 7-4.

A preliminary estimate of costs for each phase can be summarized as follows:

.1 Phase 1

Drilling, sampling and monitoring of three holes Phalen Seam (Table 7-2)	\$ 96 400
---	-----------

.2 Phase 2

Drilling sampling and monitoring of further three holes Phalen Seam. Experience gained in Phase 1 could change the cost of this phase.	96 400
--	--------

.3 Phase 3

Hydrofracturing Tests (Tables 7-3 and 7-4)	289 000
--	---------

The total Phalen Program Schedule is given in Figure 7-6.

AR-124

TABLE 7-1
RESERVOIR AND TREATMENT DATA

<u>Reservoir Data</u>	<u>Lingan Colliery</u>	<u>Phalen Colliery</u>
Coal Thickness	1.7 m to 2.0 m	1.7 m to 2.5 m
Reservoir Pressure	?	?
Cleat Direction	N 70° - 85° E	N 70° - 85° E
Depth	-650 m to -700 m	-300 m to -450 m
Minimum Stress Gradient in Overlying Strata	28 - 33 kPa/m	23 - 27 kPa/m
<u>Treatment Data</u>		
Rate	1.4 m ³ /min	1.4 m ³ /min
Expected Injection Pressure	13 to 18 MPa	8 to 12 MPa
Fluid Type	Gelled Water	Gelled Water
Viscosity	70 mPa.s	70 mPa.s
Volume	20 m ³	20 m ³
Proppant	Optional	Optional

AR-124

TABLE 7-2
PHALEN PHASE I
COST ESTIMATE

Drilling

Site Preparation 75 m @ \$100/m	\$	7 500	
Mine Services		7 500	
Drilling 3 holes (@ 50 m to 100 m)		15 000	
Provision for stuffing box and water trap		4 500	
Labour provision one man-month		4 000	
Sub-Total Drilling			\$ 38 500

Sampling and Testing

Core sampling	1 man-month	4 000	
Laboratory testing		5 000	
Sub-Total Sampling and Testing			9 000

Instrumentation and Monitoring

Pressure gauges	3 @ 200	600	
Flow meters	3 @ 2 500	7 500	
Recorders	2 @ 2 100	4 200	
Monitoring	1 man-month	4 000	
Sub-Total Instrumentation and Monitoring			16 300

Engineering Services 20 000

Contingency 15% 12 600

TOTAL ESTIMATE PHASE I \$ 96 400

AR-124

TABLE 7-3
PHALEN PHASE 3
ESTIMATED COST OF INITIAL
HYDROFRACTURING TEST

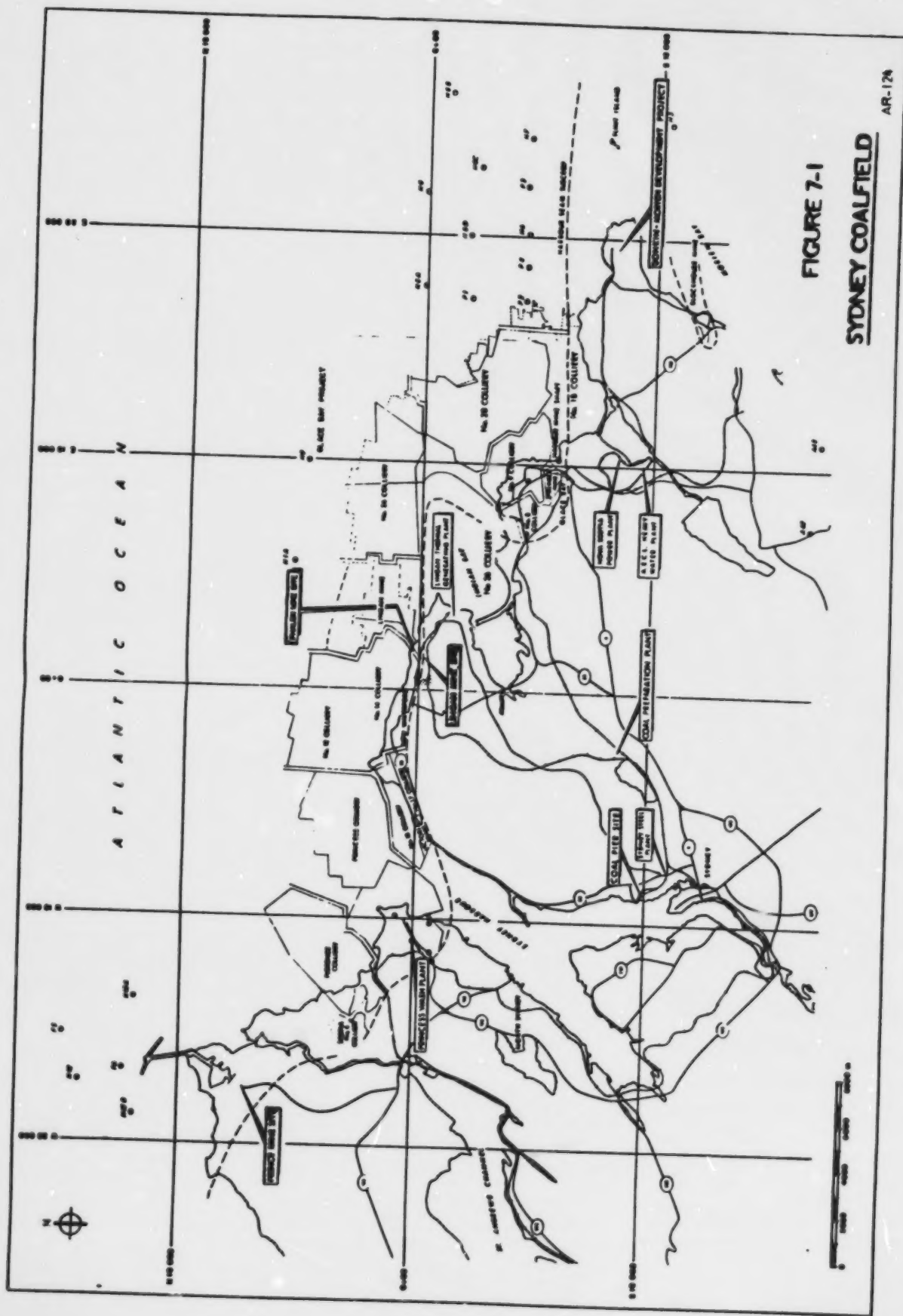
Operational

Hydrofracturing Engineer	4 man months	\$ 25 000
Laborer \$26/hr = \$4,000 month	4 man months	16 000
Supply Provision Fracturing Gell		10 000
Packers, etc.		<u>14 000</u>
		\$ 65 000
Contingency 15%		<u>10 000</u>
Sub-Total Labour and Supplies		\$ 75 000
Equipment Cost Estimate (See Table 7-4)		<u>214 000</u>
TOTAL ESTIMATE PHASE 3		<u><u>\$ 289 000</u></u>

TABLE 7-4
ESTIMATED COST OF HYDROFRACTURING EQUIPMENT

1. <u>Pumping Unit</u>		
A. GE motor (600 hp)	\$ 50 000	
B. Eaton AT-1202 two speed transmission	4 000	
C. BJ PCII LFE 6" plungers	81 250	
D. Skid construction, etc.	<u>18 750</u>	
Sub-Total Pumping Unit		\$ 154 000
2. <u>Tanks</u>		
2 Tanks:		
Material	11 200	
Labour	<u>16 000</u>	
Sub-Total 2 Tanks		28 000
3. <u>Booster Units</u>		
2 Booster Units		12 500
Contingency 10%		<u>19 500</u>
ESTIMATED COST OF HYDROFRACTURING EQUIPMENT		<u><u>\$ 214 000</u></u>

Note: Quotations and estimates received in U.S. dollars have been converted at Cdn. \$1.25.



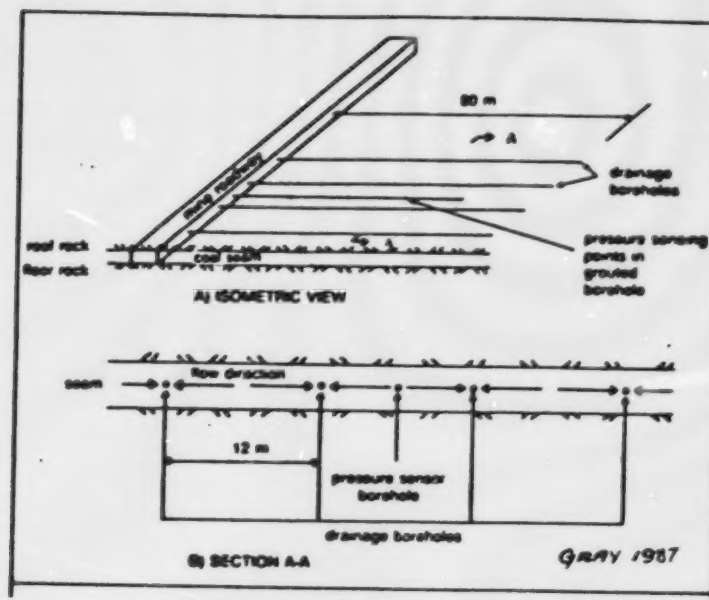
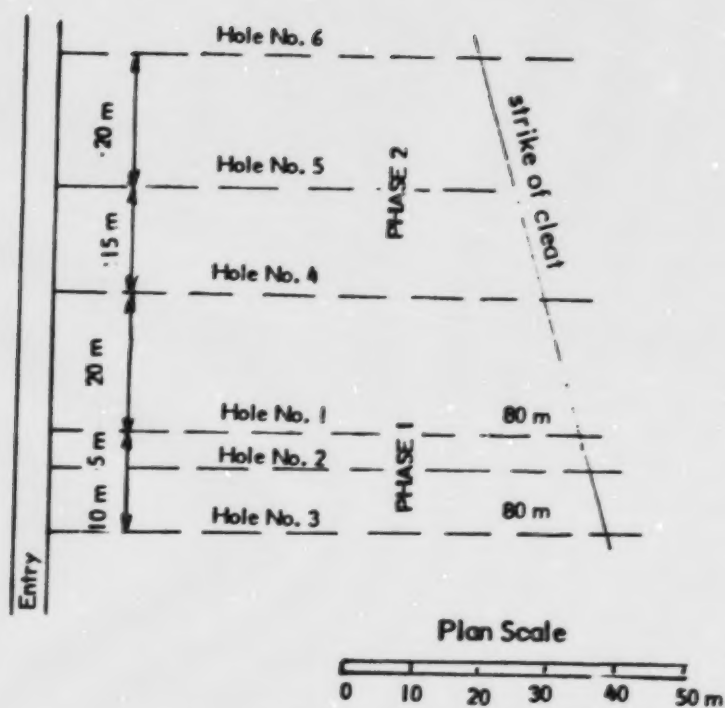


FIGURE 7-2
Isometric View of Four-Hole Drainage Trial
after Gray, 1987

AR-124



Distance between Holes 2 and 3 may be increased if open cleats found in Hole No. 1

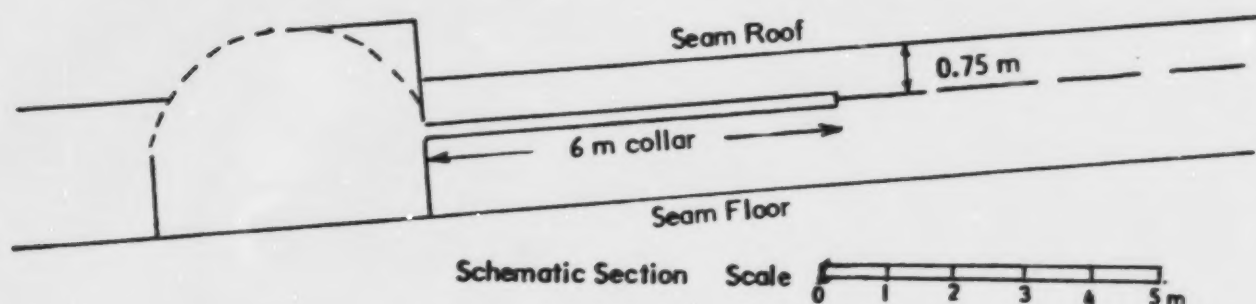


FIGURE 7-3
Drillhole Layout Test Program

AR-124

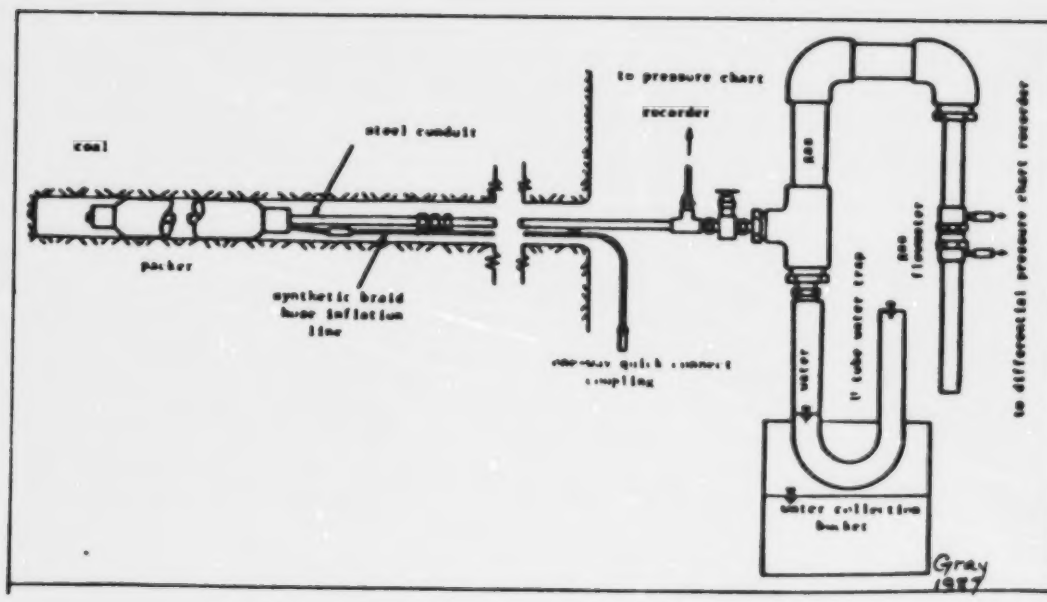


FIGURE 7-4
Pressure Buildup Test Equipment
after Gray, 1987

PHALEN PHASE 1 — TEST PROGRAM SCHEDULE

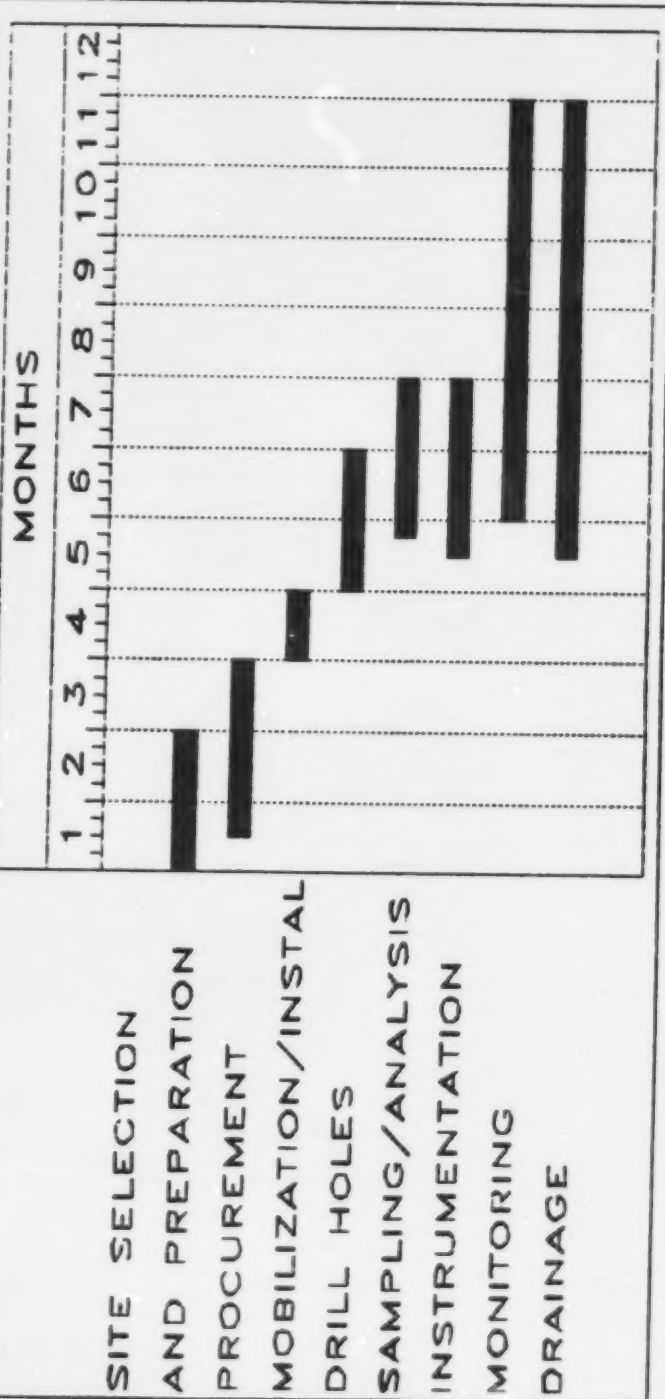
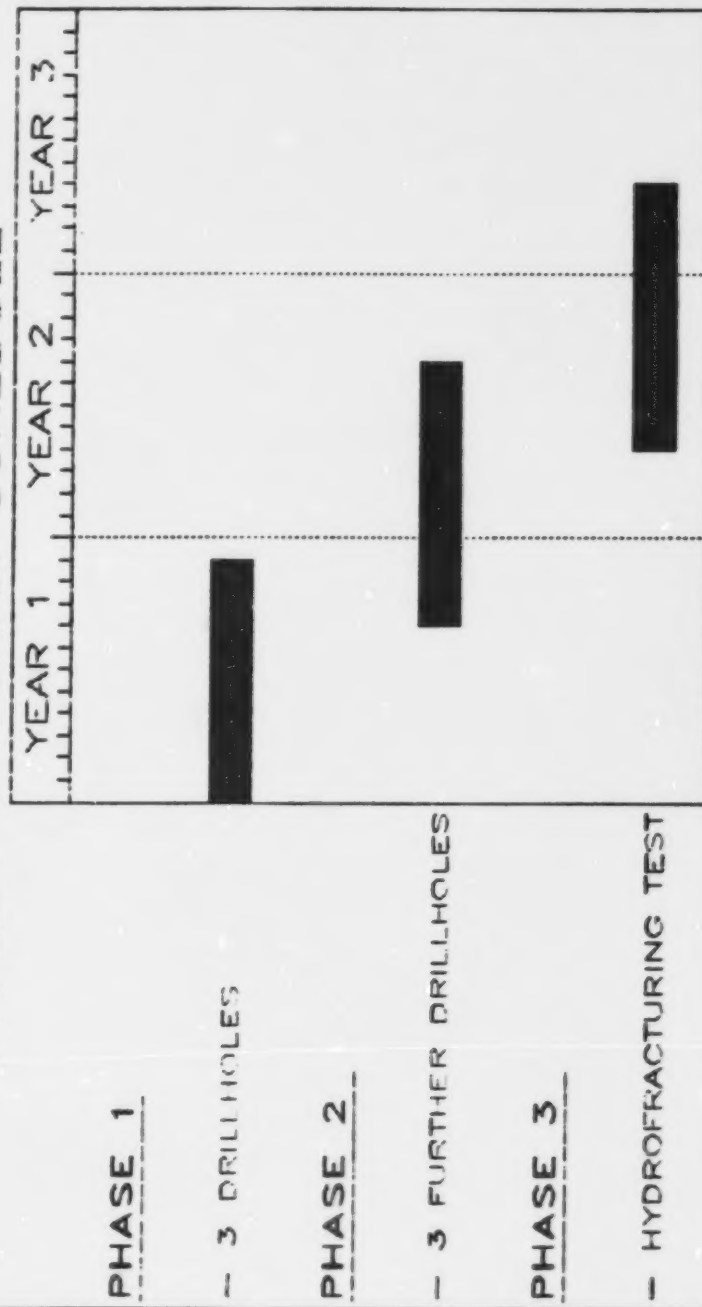
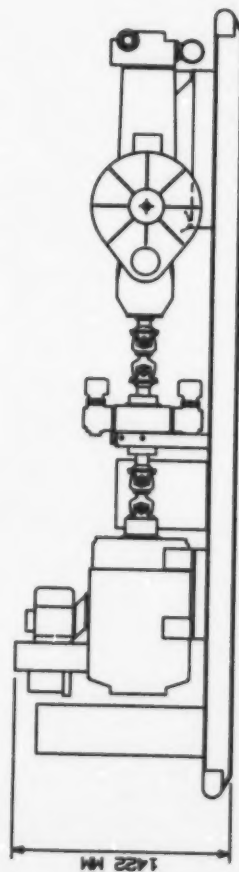
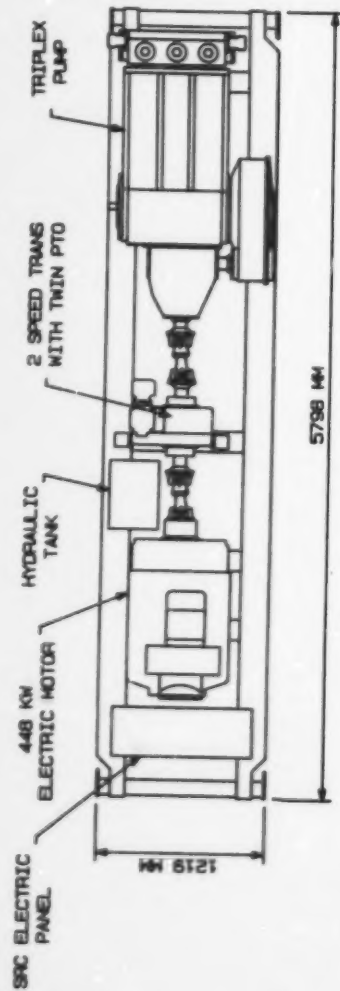


FIGURE 7-6
PHALEN PROGRAM SCHEDULE





APPROXIMATE WEIGHT 7030 KG

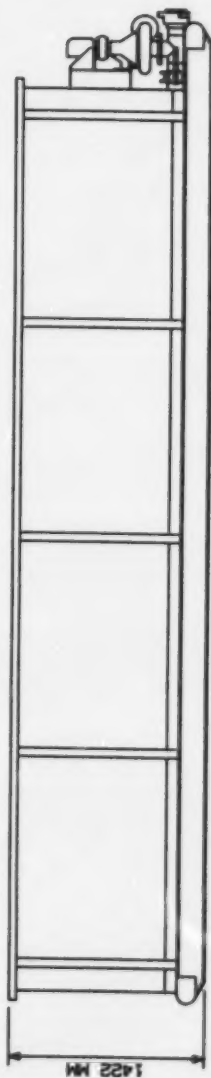
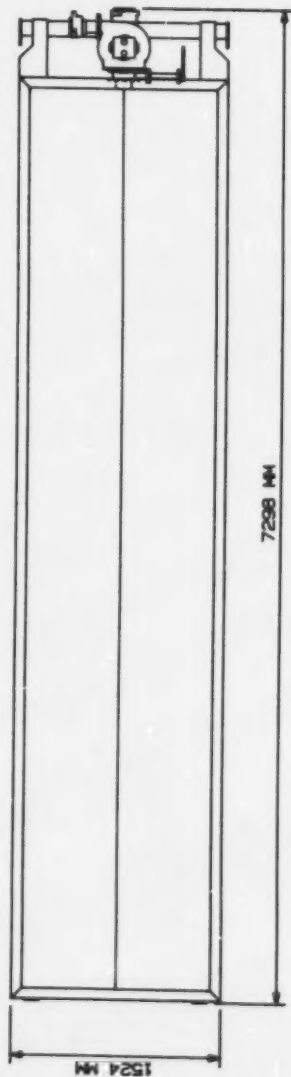
BASE MODEL SHOWN. OPTIONAL EQUIPMENT MAY VARY IN APPEARANCE.
DIMENSIONS ARE APPROXIMATE AND MAY VARY ACCORDING TO FINAL DESIGN



THIS DRAWING AND THE INFORMATION
DISCLOSED HEREIN IS THE EXCLUSIVE PROPERTY
OF NOWSCO WELL SERVICE LTD. AND IS NOT TO
BE REPRODUCED OR USED TO MANUFACTURE
ITEMS FOR USE OR SALE BY ANY OTHER PERSON
OR CONCERN WITHOUT THE WRITTEN PERMISSION
OF NOWSCO WELL SERVICE LTD.

FIGURE 7-7

DRN	PHH	CHKD	APP
DATE	20MAR08	DATE	
 NOWSCO WELL SERVICE LTD			
TITLE: PUMP SKID			
PROPOSAL			
DWG NO.	SKB1592	REV	CODE
SCALE:	N/A	SHT	1 OF 1



NOMINAL TANK CAPACITY 10M³
 APPROXIMATE HEIGHT 2630 KG

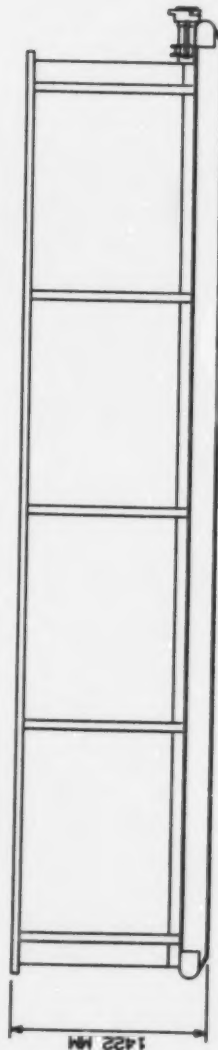
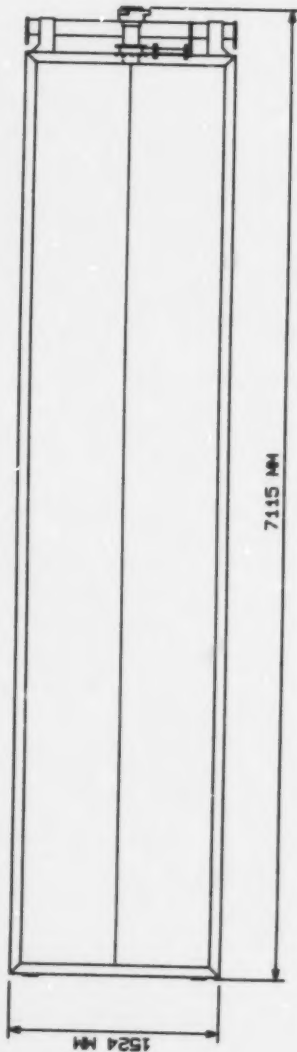


BASE MODEL SHOWN. OPTIONAL EQUIPMENT MAY VARY IN APPEARANCE.
 DIMENSIONS ARE APPROXIMATE AND MAY VARY ACCORDING TO FINAL DESIGN

THIS DRAWING AND THE INFORMATION
 DISCLOSED HEREIN IS THE EXCLUSIVE PROPERTY
 OF NOWSCO WELL SERVICE LTD. AND IS NOT TO
 BE REPRODUCED OR USED TO MANUFACTURE
 ITEMS FOR USE OR SALE BY ANY OTHER PERSON
 OR CONCERN WITHOUT THE WRITTEN PERMISSION
 OF NOWSCO WELL SERVICE LTD.

FIGURE 7-8

DRN	PHH	CHKD	APP
DATE	29MAR88	DATE	
 NOWSCO WELL SERVICE LTD			
TITLE		BATCH TANK 10 M ³	
		PROPOSAL	
DWG NO.	SKB1593	REV	CODE
SCALE	N/A	SHT	1 OF 1



NOMINAL TANK CAPACITY 10M³
 APPROXIMATE WEIGHT 2510 KG

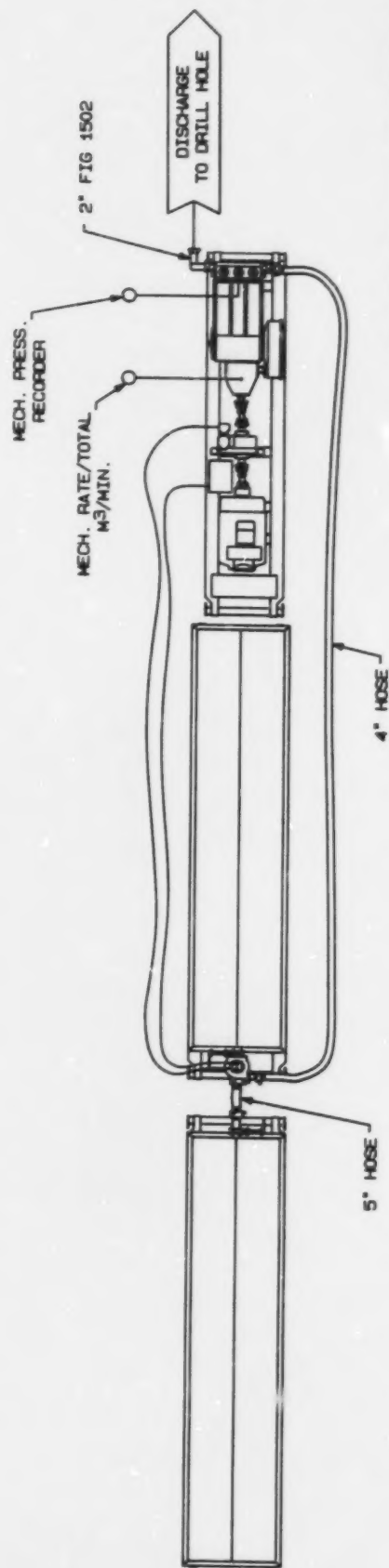
BASE MODEL SHOWN. OPTIONAL EQUIPMENT MAY VARY IN APPEARANCE.
 DIMENSIONS ARE APPROXIMATE AND MAY VARY ACCORDING TO FINAL DESIGN



THIS DRAWING AND THE INFORMATION
 DISCLOSED HEREIN IS THE EXCLUSIVE PROPERTY
 OF NOWSCO WELL SERVICE LTD. AND IS NOT TO
 BE REPRODUCED OR USED TO MANUFACTURE
 ITEMS FOR USE OR SALE BY ANY OTHER PERSON
 OR CONCERN WITHOUT THE WRITTEN PERMISSION
 OF NOWSCO WELL SERVICE LTD.

FIGURE 7-9

DWG	PHH	CHKD	APP
DATE	06APR88	DATE	DATE
 NOWSCO WELL SERVICE LTD			
TITLE: BATCH TANK 10 M ³ PROPOSAL			
DWG NO. SKB1602		REV CODE	
SCALE: N/A		SHT 1 OF 1	



THIS DRAWING AND THE INFORMATION
 DISCLOSED HEREIN IS THE EXCLUSIVE PROPERTY
 OF NOWSCO WELL SERVICE LTD. AND IS NOT TO
 BE REPRODUCED OR USED TO MANUFACTURE
 ITEMS FOR USE OR SALE BY ANY OTHER PERSON
 OR CONCERN WITHOUT THE WRITTEN PERMISSION
 OF NOWSCO WELL SERVICE LTD.

FIGURE 7-10

POWER SUPPLY: 230/450 VAC
 TRIPLEX RATE & PRESS. 18 LITERS/SEC @ 17,260 Kpa
 CENTRIFUGAL RATE & PRESS. 18 LITERS/SEC @ 414 Kpa

BASE MODEL SHOWN. OPTIONAL EQUIPMENT MAY VARY IN APPEARANCE.
 DIMENSIONS ARE APPROXIMATE AND MAY VARY ACCORDING TO FINAL DESIGN

DRN	PHH	CKD	APP
DATE	30MAR88	DATE	DATE


 NOWSCO WELL SERVICE LTD

TITLE: HYDRAULIC FRACTURING
 EQUIPMENT LAYOUT

DWG NO.	SKB1594	REV CODE
SCALE:	N/A	SHT 1 OF 1

PRE-DRAINAGE IN COAL MINES

SECTION 8 - REFERENCES / SELECTED BIBLIOGRAPHY

- Anon.
1986: In Seam Gas Drainage Reduces Risk of Underground Explosion; Aust. Coal Miner (Jan), p. 22-23.
- Baker, E.C.; Oyler, D.C.; Perry, J.H. and Finfinger, G.L.
1983: Economic Evaluation of Directional Drilling for Methane Drainage from Coalbeds; U.S.Bu. Mines, R.I. 8842
- Battino, S. and Hargraves, A.J.
1982: Seam Gas Drainage Experiments in Some Collieries of B.H.P. p. 157 - 168, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with particular reference to the Working Seam; Aust. I.M.M. Wollongong, 1982.
- Blauer, R.E. and Kolhaas, C.A.
1974: Formation Fracturing with Foam; SPE Paper 5003, 49th Ann. Fall Mtg. (Oct), Houston, Texas.
- Boxho, J., et al
1980: Firedamp Drainage: Handbook for the Coal Mining Industry in the European Community; Glueckauf, Essen, 415 p.
- Boyer II, C.M.; Stubbs, P.B. and Schwerer, F.C.
1986: Measurement of Coalbed Properties for Hydraulic Fracture Design and Methane Production; p. 591 - 598, in Proc: Unconventional Gas Technology Symposium of the Society of Petroleum Engineers, Louisville, Kentucky; May, 1986 SPE Paper No. 15258
- British Coal - Headquarters Technical Department
1987: Guided Longhole Drilling; HQ Tech. Dep't. Brochure, 2 p.
- Cervik J.; Fields H.H. and Aul, G.H.
1975: Rotary Drilling Holes in Coalbeds for Degasification U.S.Bu. Mines, R.I. 8097
- Commission of the European Communities
1980: Information Symposium "Methane, Climate, Ventilation in Coalmines of the European Communities; Luxemburg, Nov., 1980, Colliery Guardian EUR6980e
- Cowan, J; O'Gorman, G.R. and Hemphill, P.
1987: The Devran/Shell Oil Mining Project; Paper No. 165 presented at C.I.M. Ann. Gen. Mtg Toronto, May.

Creedy, D.P.

- 1986: Methods for the Evaluation of Seam Gas Content from Measurements on Coal Samples; Min. Sc. and Tech., No. 3, p. 141-160.

Curl, S.J.

- 1978: Methane Prediction in Coal Mines; IEA Coal Research, Report No. ICTIS/TR 04, London.

Dames & Moore and Stephenson, H.G.

- 1977: A Study of Methane Drainage Practices in Coal Mines and Their Potential Application to Canadian Conditions; DSS Contract for CANMET Report No. 05Q76-00139.

Deul, M.

- 1979: The U.S. Bu.Mines Program for Methane Drainage to Reduce Safety Hazards in Coal Mines; Proc. 2nd Ann. Symp. Methane Recovery from Coal Beds, April.

Diamond, W.P.

- 1978: Evaluation of the Methane Gas Content of Coalbeds: Part of a Complete Coal Exploration Program for Health and Safety and Resource Evaluation; 2nd Int. Coal Expl. Symp; Denver, CO. (Oct.) Proc. pp 211-227.

Diamond, W.P. and Levine, J.R.

- 1981: Direct Method Determination of the Gas Content of Coal: Procedures and Results; 36 p. U.S.Bu. Mines, RI 8515

Diamond, W.P. and Oyler, D.C.

- 1987: Effects of Stimulation Treatments on Coalbeds and Surrounding Strata; U.S.Bu. Mines, RI 9083, 48 p.

Dunmore, R. and Creedy, D.P.

- 1980: Measuring Methane in Coal Seams and Predicting its Rate of Emission; p. 148-169 in Proc. Inf. Symp. "Methane, Climate, Ventilation in the Coalmines of the European Communities"; Luxembourg, Nov., 1980, Colliery Guardian

Edling, L.

- 1985: Report - Hole Drilled with a In-hole Motor in a Coal Seam in Lingan Phalen Mine; Internal Report Boyles Bros. Drilling Co. to C.B.D.C.

English, L.M.

- 1984: Pressure and Temperature Correction for Coal Desorption Measurements; SME-AIME Preprint 84-388

Finfinger, G.L.; Perry, J.H. and King, R.L.

- 1982: Experience with Horizontal Boreholes in a Western Coalbed; p. 105-111, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollangong, 1982.

Fisekci, M.Y. and Barron, K.

- 1975: Methane Pressure and Flow Measurements in Coal and Surrounding Strata; Canadian Inst. Min. and Met. Bulletin, (Oct.)

Gray, I.

- 1982: A Study of Seam Gas Drainage in Queensland; p. 218-229, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

- 1987a: Reservoir Engineering in Coal Seams Part 1 - The Physical Process of Gas Storage and Movement in Coal Seams; SPE Reservoir Engineering, (Feb), p 28-34.

- 1987b: Reservoir Engineering in Coal Seams Part 2 - Observations of Gas Movement in Coal Seams; SPE Reservoir Engineering, (Feb), p 35-40.

Hargraves, A.J. (ed)

- 1982a: Proc. Symp. on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. Inst. Min. and Met., U of Wollongong, May, 1982.

- 1982b: Background to Seam Gas Drainage in Australia - Keynote Address; pp 21-34, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Hayes, P.J.

- 1982: Factors Affecting Gas Release from The Working Seam; p. 62-69, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Hebblewhite, B.K.; Leung, L. and Holt, G.E.

- 1985: A Review of Australian Developments in the Technologies and Application of In-seam Drilling and In-seam Seismics; p. 337-357 in Proc. 7th Int. Conf. on Coal Research, Pretoria, R.S.A., October, 1985; Vol. 2.

Hebblewhite, B.K.; Richmond, A.J. and Allan, J.

- 1982: Australian Experience in Long-Hole In-seam Drilling Technology For Seam Gas Drainage; p. 202-216, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Hextall, J.L.

- 1983: Underground Guided Longhole Drilling (GLHD) Trials at H60's - Cotgrave Colliery; NCB, MRDE Tunneling and Transport Branch, Tech. Memorandum No: TT(83)9.

Hicks, L.

- 1986: Review of Gas Supply From Lingan, No. 26 and Phalen Mines;
CBDC Internal Report

Highton, W.

- 1982a: Experience With Pre-drainage of Seam Gas in the Western Area of the National Coal Board; p. 124-139, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Highton, W.

- 1982b: Retreat Mining Methane Drainage; p. 268-282, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Higuchi, K.; Ohga, K. and Isobe, T.

- 1982: Considerations to Increase Gas Drainage Ratio Basing Upon Methane Drainage Practice in Japan; p. 232-238, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Houseknecht, D.W.

- 1982: Probabilities of Encountering Coalbed Discontinuities During Vertical and Horizontal Borehole Drilling; U.S.Bu. Mines, R.I. 8665

Hubbert, M.K. and Willis, D.G.

- 1957: Mechanics of Hydraulic Fracturing; Trans. of SPE of AIME Vol. 210, p. 153-168.

Jones, A.H.; Ahmed, U.; Abou-Sayed, A.S.; Mahyera, A. and Sakashita, B.

- 1982: Fractured Vertical Wells Versus Horizontal Boreholes for Methane Drainage in Advance of Mining U.S. Coals; p. 172-198, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Jones, A.H.; Ahmed, U.; Bush, D.D.; Kelkar, S.M.; Rakop, K.C.; Holland, M.T.; Bowman, K.C. and Bell, G.J.

- 1984: Experience With Gas Recovery from Unmineable Coal in the San Juan Basin; 14 p., Preprint: 1984 International Gas Research Conference.

Jones, A.H.; Bell, G.J. and Morales, R.H.

- 1986: Coalbed Hydraulic Fracture Treatment Empirical Relationships and Computer Simulation; p. 699-709, In Proc: Unconventional Gas Technology Symposium of Society of the Petroleum Engineers, Louisville, Kentucky; (May), SPE Paper No. 15242

Kahil, A.A. and Masszi, D.

- 1982: Technical Considerations for the Design of a Demethanization Program; p. 99-104, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

- Kelly, J. and England, R.
1987: Development of the Automatic Go-60 Drill; CIM Bull. (July), p. 71-74.
- King, R.L. and Trevits, M.A.
1982: Methane Drainage: Development of a Large-Scale Field Demonstration Using Vertical Boreholes; p. 81-87, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.
- Korfmann, Heinz-Diether, et al.
1986: Air Cooling and Methane Drainage in the Federal Republic of Germany; World Mining Equipment, July, 1986.
- LeBlanc, W.
1977: Development of Methane Drainage in Cape Breton Coal Mines; Internal Report C.B.D.C., 9 p.
- Lama, R.D. and Bartosiewicz, H.
1982: Determination of Gas Content of Coal Seams; p. 36-49, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.
- Lambert, S.W. and Trevits, M.A.
1979: Methane Drainage Ahead of Mining Using Foam Stimulation Mary Lee Coalbed, Alabama; Dep't of Energy RI PMTC-3-(79), January.
- Lambert, S.W.; Trevits, M.A. and Steidl, P.F.
1980: Vertical Borehole Design and Completion Practices to Remove Methane Gas From Mineable Coalbeds; U.S. Dept'. Energy, Rpt. No. DOE/CMTC/TR-80/2, Aug.
- Layne, A.W.
1986: Coalbed Methane Stimulation Analysis; Proc. Unconventional Gas Recovery Contractors Meeting, Morgantown, W. Virg., (Nov. 1985) p. 91-98
- Lingard, P.S.; Phillips, H.R. and Doig, I.D.
1982: The Permeability of Some Australian Coals; p. 70-79, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.
- Logan, T.L.; Seccombe, J.C. and Jones, A.H.
1986: Hydraulic Fracture Stimulation Results and Diagnostics in Deeply Buried Coal Seams, Piceance Basin, Colorado; Proc. 27th U.S. Symp. Rock Mech: Key to Energy Production, Tuscaloosa, AL; Soc. Mining Eng.; Ch 94, p. 677-681.

Logan, T.L.; Seccombe, J.C. and Jones, A.H.

- 1986: Hydraulic Fracture Stimulation and Openhole Testing of a Deeply Buried Coal Seam in the Piceance Basin, Colorado; p. 501-512, In Proc: Unconventional Gas Technology Symposium of the Society of Petroleum Engineers Louisville, Kentucky, SPE Paper No. 15251.

Lunarzewski, L.W.

- 1982: Methane Drainage Effectiveness With Regard to Different Mining Systems in Polish Coal Mines; p. 283-298, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Mahoney, J.V.; Stubbs, P.B.; Schwerer, F.C. and Dobscha, F.X.

- 1981: Effects of a No-Proppant Foam Stimulation Treatment on a Coal Degasification Borehole; J. Pet. Tech., Vol. 33/11, p. 2227-2235.

Marshall, P.; Lama, R.D. and Tomlinson, E.

- 1982: Experience on Pre-drainage of Gas at West Cliff Colliery; p. 141-155, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Mavko, B.B.; Hanson, M.E.; Nielsen, P.E. and Logan, T.L.

- 1986: Hydraulic Fracture Model for Application to Coal Seams; Proc. 27th U.S. Symp. Rock Mech: Key to Energy Production, Tuscaloosa, AL.; Soc. Mining Eng.; Ch. 103, p. 741-748.

Morris, I.H.

- 1982: Firedamp Drainage Installations; p. 306-317, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Noack, K.

- 1982: Pre-drainage of Methane in West German Coal Mines; p. 113-117, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Nolte, K.G.

- 1986: A General Analysis of Fracturing Pressure Decline with Application to Three Models; Soc. Petrol. Eng. Formation Evaluation, Dec., p. 571

Norton Canada Inc.

- 1984: Norton Christensen Double Tilted U-Joint Navigation Sub; Norton Accu-Trac Directional Drilling Services, Company Brochure, 9 p.

Norwest Resource Consultants Ltd.

- 1985: A Detailed Review and Evaluation of Methane Prediction Techniques and Their Applicability to Canadian Coal Mines; CANMET/DSS File No. 075Q.23440-5-9008.

Oyler, D.C. and Diamond, W.P.

1982: Drilling a Horizontal Coalbed Methane Drainage System From a Directional Surface Borehole; U.S.Bu. Mines, R.I. 8640

Patching, T.H.

1965: Variations in Permeability of Coal; Department, Mining and Metallurgy, University of Alberta, Edmonton, Alberta.

Perry, J.H.; Prosser, L.J. Jr., and Cervik, J.

1982: Degasification of the Mary Lee Coalbed, Brookwood, Alabama
U.S.Bu. Mines, R.I. 8669, 1982.

Price, H.S.; McCulloch, R.C.; Edwards, J.C. and Kissel, F.N.

1973: A Computer Model Study of Methane in Coal Beds; CIM Bull., Vol. 66, Sept., p. 103-112.

Prosser, L.J. Jr., Finfinger, G.L. and Cervik, J.

1981: Methane Drainage Study Using an Underground Pipeline, Marianna Mine 58; U.S.Bu. Mines, R.I. 8577.

Schwoebel, J.J.

1987: Coalbed Methane: Conversion of a Liability to an Asset; Min. Eng., Vol. 39, No. 4, p. 270-274.

Steidl, P.F.

1978: Foam Stimulation to Enhance Production from Degasification Wells in the Pittsburgh Coalbed; U.S.Bu Mines R.I. 8286.

Stewart, W.J. and Barro, L.

1982: Coal Seam Degasification by use of Hydraulic Fracturing in Vertical Wells: Case Histories; p. 89-97, in Hargraves, A.J. 1982: Proc Symp on Seam Gas Drainage with Particular Reference to The Working Seam; Aust. I.M.M., Wollongong, 1982.

Suttil, K.

1987: Deflection Drilling; Eng. & Mining Journal, Vol. 188/7, p. 48-51.

Thakur, P.C. and Dahl, H.D.

1982: Horizontal Drilling - A Tool for Improved Productivity; Mining Engineering (March) pp. 301-304.

Thakur, P.C. and Poundstone, W.N.

1980: Horizontal Drilling Technology for Advance Degasification; Mining Engineering June, 1980.

Thompson, D.S.

1984: High-Angle Drilling for the Centralia Partial-Seam Crip Test; Lawrence Livermore National Laboratory, Report UCRL 53508.

Treskow, A. von and Fitzner, G.

1986: Die Ergebnisse der Grubenabsaugung im deutschen Steinkohlenbergbau in den Jahren 1984/85; Glueckauf, Vol. 122, No. 19, p. 1269-1271.

von Schonfeldt, A.; Pethini, B.R. and Aul, G.N.

1980: Advance Methane Control and its Impact on Gas Emissions in the Pocahontas #3 Coal Seam; p. 347-354, In Proc: Unconventional Gas Technology Symposium of the Society of Petroleum Engineers Louisville, Kentucky; May, 1980.
SPE Paper No. 8969.

Williams, R.J.; Allan, J.; Truong, D. and Beamish, B.B.

1985: An Evaluation of Seam Gas Drainage by Long-hole Drilling; 21st Int. Conf. of Safety in Mines Res. Inst. Sydney, October, 1985.

AR-124

APPENDIX A

DETERMINATION OF GAS CONTENT
OF COAL SEAMS

LAMA, R.D. AND BARTOSIEWICZ, H.; 1982

DETERMINATION OF GAS CONTENT OF COAL SEAMS

By

R.D. Lana¹ and H. Bartosiewicz²

ABSTRACT

A method for the determination of the gas content of coal seams is presented which improves upon existing methods. The method makes use of a gravitational technique for measurement of the gas desorbed. The amount of lost gas is estimated by determining an empirical relationship relating the desorption of gas with time for each sample separately. This is to replace the assumption that desorption follows a square root law with time. Laboratory investigations have been conducted which indicate that the square root desorption law does not hold good for carbon dioxide. Results of investigations on various factors influencing the technique are presented. It is shown that co-efficients of the empirical equation are dependent upon the elapsed time and the period of observation. Data about gas pressure measurements is presented and it is concluded that gas pressure can be approximated to 0.6 times the pressure below a column of water equal to the depth of cover. This estimate is good enough for adsorption tests to evaluate desorption co-efficients. The technique for repressurisation of the sample,

along with the equipment required, is described.

INTRODUCTION

The determination of the gas content of coal seams is of fundamental importance in the planning of ventilation systems in coal mines. Its importance has increased very rapidly over the last 10 years, mainly due to developments in highly productive longwall equipment. Modern equipment capable of producing up to 1 000 tonnes per hour, requires that the environmental conditions, both gas and dust, be kept within specified limits.

The effect of specific methane emission (m^3/t) on the production potential of the working face is given in Fig. 1. It is obvious that in the planning of any operation in gassy mines, specific gas emissions for the area must be known in advance. For example, investigations on gas emission at West Cliff Colliery were conducted before the introduction of longwall mining. It was concluded that gas emission from the Bulli Seam must be reduced by at least 50%, if production of 6 000 - 7 000 tonnes per day is to be achieved.

Estimation of gas content is not only essential for ventilation, but also permits assessment of outbursts of gas and coal hazards in advance. It helps in the planning of suitable mining methods, selection of

¹Manager Technology, Rambla Coal & Coke Pty. Limited, Wollongong, New South Wales.
(Previously C.S.I.R.O., Division of Applied Geomechanics, Syndal, Victoria.)

²C.S.I.R.O., Division of Applied Geomechanics, Syndal, Victoria.

mining equipment and layout to combat these conditions. Gas content estimation, as one of the important parameters to be measured in the early stages of exploration, is becoming rapidly accepted.

METHODS OF ESTIMATING THE GAS CONTENT OF COAL SEAMS

Some of the earliest attempts to measure the gas content of coal were made by Graham, (1937-8), but intensive investigations really began in the mid-sixties with developments in the U.K., Germany, France, Belgium and the U.S.A.

Methods for the estimation of the gas content of coal seams can be grouped into two categories:

(i) Direct

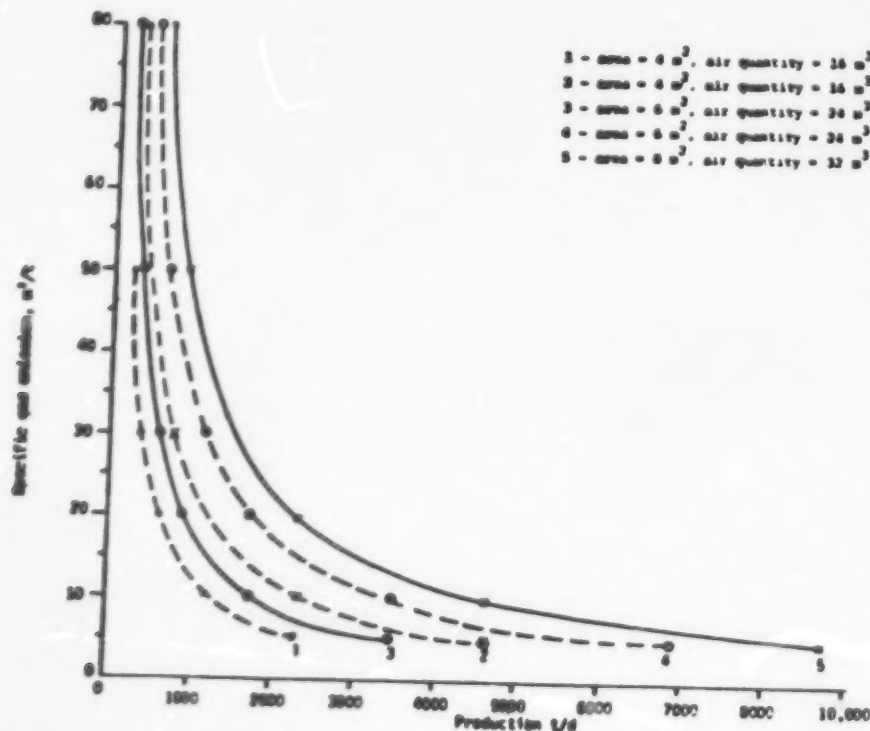


Fig. 1. Effect of specific gas emission on maximum production from a longwall face: % CH_4 concentration 1.25%; Coefficient of irregular emission 1.5.

(ii) Indirect

Direct methods are based upon extracting a coal sample, enclosing it in a sealed chamber and measuring the gas evolved from it. Indirect methods are based upon either gas adsorption characteristics of coal under given pressure and temperature conditions, or other empirical data obtained from existing mines which relate the gas content of coal to certain other related parameters. A summary of the various methods is given in Table 1.

DIRECT METHOD

This method was first developed on a scientific basis by CERCHER and has since been used in France, Belgium, Germany and the U.K.

Table 1. Methods of estimation of gas content of coal seams

Method	Requirements	Methods of interpretation	Accuracy	Country	Method of measurement	Applicability
<u>DIRECT</u>	Cores	Requires knowledge of gas emission laws.	Depending upon time lost, core quality and applicability of gas emission laws. $\pm 50\% - \pm 20\%$	U.K. U.S.A. Australia	Volumetric gas emission at atmospheric pressure followed by crushing.	Good from surface boreholes and doubtful from underground boreholes, fractured cores etc. Degree of fracturing influences results.
<u>DIRECT</u>	Sampling of fractions.	Empirical	?	Poland	Volumetric gas emission.	Existing mines.
<u>Indirect Statistical</u>	Lumps of coal	Statistical analysis of gas content of lumps.	$\pm 20\%$, statistically measured values from face samples are consistently higher.	U.K.	Sample (30-40 mm size), collection at the face and estimation of gas contents.	For seams under mining, face sampling, applicability to highly variable and high rank coals not proved.
<u>Indirect Adsorption isotherms</u>	Pressure measurement and sampling.	Direct reading for adsorption isotherms.	Depending upon pressure measurements and sampling. $\sim 10-20\%$	Poland USSR Australia Germany	Volumetric techniques and gravimetric techniques.	For seams under mining and seams lying above or below.
<u>Indirect Chemical analysis</u>	Proximate analysis of coal.	Empirical	$\sim 30-100\%$	USSR U.S.A. Poland	Chemical analysis	New and existing mines.
<u>Indirect</u>	Exhaust Ventilation sampling	Subtraction techniques	$\pm 20\%$	Australia	CH ₄ or CO ₂ gas analysis (1).	Existing mines.
<u>Indirect</u>	Sampling of exhaust mine air	Empirical	$\pm 50\%$	Germany U.S.A.	Volumetric gas emission.	Existing mines - general make of gas.

The original method consists of sampling coal seams by drilling and enclosing the cuttings in a container as soon as possible from which the gas content Q_2 is measured volumetrically (Bertard et al., 1970; Belin, 1971).

The sample is then crushed in a sealed chamber and additional gas evolved (Q_3) is measured. Crushing is done at atmospheric pressure (Germany, France, Belgium, Poland and the U.S.A.), or at zero partial pressure (U.K.). The amount of gas lost during extraction and transfer (Q_1) is calculated using the relationship

$$Q_1 = K(t)^n \quad (1)$$

The value of n is either calculated from desorption data or assumed. The French (Bertard et al., 1970) and the U.S.A. investigators (Kissel et al., 1973) believe this to be 0.5; U.K. (Airey, 1968), believe that its value varies between 0.25 - 0.3, and Germans, between 0.3 and 0.35. The error in the value of Q_1 , however, by using $n = 0.3$ instead of 0.5 will be almost 100%. The total gas content is then given by

$$Q = Q_1 + Q_2 + Q_3 = a^3/t \quad (2)$$

Certain modifications of the direct method have been tried. These include sealing of the sample in situ (Ettinger et al., 1958; Lidin et al., 1965), and sealing of the core barrel (Tarnowski, 1960). Some investigators believe that the methods of direct determination are not accurate enough to warrant the calculation of lost gas and instead assume this quantity to be 10% of $(Q_2 + Q_3)$.

The size and shape of the sample is an important consideration in the direct method and it is here that conflicting views exist.

The authors believe that it is imperative to obtain full cores to determine accurately the gas content. The French (CERCHER) believe that it is sufficient to obtain cuttings of 2mm size and above with a total mass of 10g. In the U.K., a sample of 30g is supposed to be sufficient to give the required accuracy.

INDIRECT METHOD

The indirect method has many variations. These include:

- (a) adsorption techniques where adsorption isotherms are obtained in the laboratory and pressure measurements are done in the field, and
- (b) empirical techniques based on estimation of the gas content from certain indirect measurements where the correlation has already been established between these indirect measurements and the gas content of a coal seam or an area.

Measurement of pressure in the field presents problems in highly porous seams where the pressure gradients are low and the length to which holes must be drilled is long, particularly when measurement is done from underground. Sealing of holes, length of seals and measurement time are some of the problems that need to be satisfactorily overcome before these measurements can be made. Methods are available to solve these problems, and appropriate answers have been found and tried with success.

A number of indirect methods exist and have been used to estimate the gas content of coal seams. These include exhaust ventilation sampling (Lams, 1980); sampling from ribsides (Tarnowski, 1972); use of

empirical data relating the gas emitted into ventilating air and the gas content of the coal (Noak, 1976); proximate analysis and gas adsorption correlation (Ettinger et al., 1958; Kiss, 1977).

DIRECT METHOD USING GRAVITATIONAL TECHNIQUES

While conducting tests using the conventional direct method, as developed by Bertard (1970) and modified by USBM, it was realised that large errors resulted in the estimation of the gas content of coal seams because of the following factors:

(i) The emission of gas from coal samples is sensitive to changes in pressure. The use of large sized sample containers to accommodate the full length of the core requires large capacity measuring cylinders. This results in subjecting the coal sample to a sudden vacuum as the sample container is connected to the measuring cylinder.

(ii) When the gas content of a coal seam is large the measuring cylinder needs to be replaced frequently. During this stage, there is a build up of pressure (above atmospheric pressure) which causes a change in the condition of gas emission from the sample (slowing down of the gas emission rate).

(iii) When cores are fractured, loss of gas in the initial stages is very high and it is advisable to close the valve of the sample container immediately on enclosing the sample. The time elapsed between the start of measuring gas (first reading) and enclosure of the sample results in a build up of positive pressure invalidating the method of analysis.

(iv) Automation of the gas measuring system is very difficult and costly.

To overcome these difficulties, the gas emission measurement technique has been modified and a new method has been used where, instead of volumetric measurements, gravimetric techniques are adopted to measure the gas emission of a sample.

The equipment used for the method is shown in Fig. 2. It consists of a sample container, a mercury seal, an electronic digital balance and a digital recorder (not shown in the figure). In case of a high moisture content of the sample (wet drilling) a moisture absorbing attachment is connected to the sampling tube.

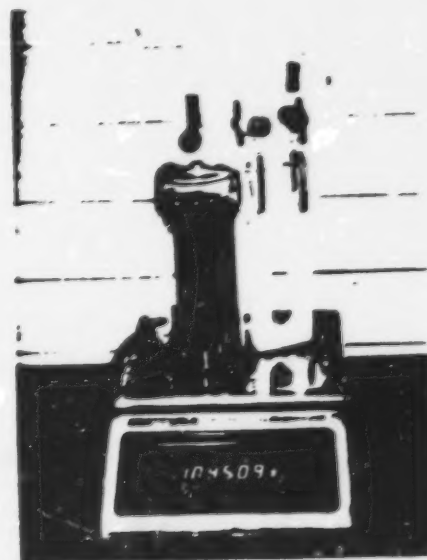


Fig. 2 Gravimetric gas adsorption cell with moisture adsorber

The accuracy of the system depends upon the balance. For coal samples up to 1.5kg weight capacity the accuracy in measurement is 1.5cc for CH_4 and 0.5cc for CO_2 . For

empirical data relating the gas emitted into ventilating air and the gas content of the coal (Noak, 1976); proximate analysis and gas adsorption correlation (Ettinger et al., 1958; Kim, 1977).

DIRECT METHOD USING GRAVITATIONAL TECHNIQUES

While conducting tests using the conventional direct method, as developed by Bertard (1970) and modified by USBM, it was realised that large errors resulted in the estimation of the gas content of coal seams because of the following factors:

(i) The emission of gas from coal samples is sensitive to changes in pressure. The use of large sized sample containers to accommodate the full length of the core requires large capacity measuring cylinders. This results in subjecting the coal sample to a sudden vacuum as the sample container is connected to the measuring cylinder.

(ii) When the gas content of a coal seam is large the measuring cylinder needs to be replaced frequently. During this stage, there is a build up of pressure (above atmospheric pressure) which causes a change in the condition of gas emission from the sample (slowing down of the gas emission rate).

(iii) When cores are fractured, loss of gas in the initial stages is very high and it is advisable to close the valve of the sample container immediately on enclosing the sample. The time elapsed between the start of measuring gas (first reading) and enclosure of the sample results in a build up of positive pressure invalidating the method of analysis.

(iv) Automation of the gas measuring system is very difficult and costly.

To overcome these difficulties, the gas emission measurement technique has been modified and a new method has been used where, instead of volumetric measurements, gravimetric techniques are adopted to measure the gas emission of a sample.

The equipment used in the method is shown in Fig. 2. It consists of a sample container, a mercury seal, an electronic digital balance and a digital recorder (not shown in the figure). In case of a high moisture content of the sample (wet drilling) a moisture absorbing attachment is connected to the sampling tube.

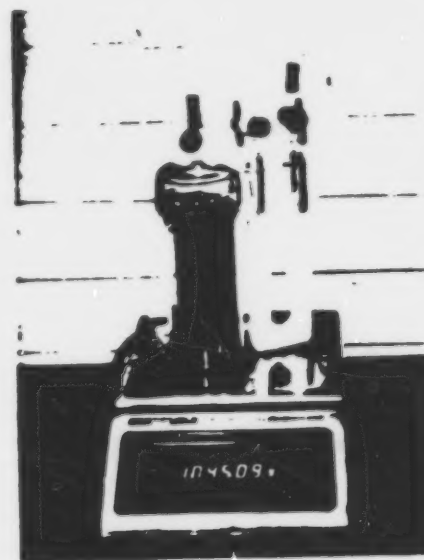


Fig. 2 Gravimetric gas adsorption cell with moisture adsorber

The accuracy of the system depends upon the balance. For coal samples up to 1.5kg weight capacity the accuracy in measurement is 1.5cc for CH_4 and 0.5cc for CO_2 . For

samples greater than 1.5kg weight, the accuracy drops because of balance limitations and is approximately 15cc for CH_4 and 5cc for CO_2 . The accuracy in gas measurement in cc/g is better than 0.007cc/g for CH_4 and 0.0025cc/g for CO_2 .

The method of measurement of the gas content is as follows:

- (i) Transfer the coal sample into the pre-cleaned and pre-weighed sample container as soon as possible.
- (ii) Record the change in weight every 30 seconds in the beginning with the time between readings extending to several hours at later stages.
- (iii) Continue gas emission tests until the rate of gas emission drops to less than 0.1cc/g/day, then transfer the sample to an adsorption bomb.
- (iv) Pressurise the adsorption bomb containing the sample to a pre-determined pressure until equilibrium is obtained, as determined by the change in weight not exceeding 0.01cc/g/day. The method of pressurisation is described in the following section. The sample is then allowed to desorb at constant temperature and the desorption rate is measured, starting within 45 seconds. The desorption data is interpolated to obtain the value of n (Eq.7) using the least square method by using data for different time intervals (60-500s; 60-1000s and 500-1000s, etc.) depending on the expected time lost in field sampling. The field curve is superimposed on the laboratory obtained curve and the value of n is obtained by matching the field curve with the laboratory curve. This value is then used in calculating the lost gas.

REPRESSURISATION OF COAL SAMPLES

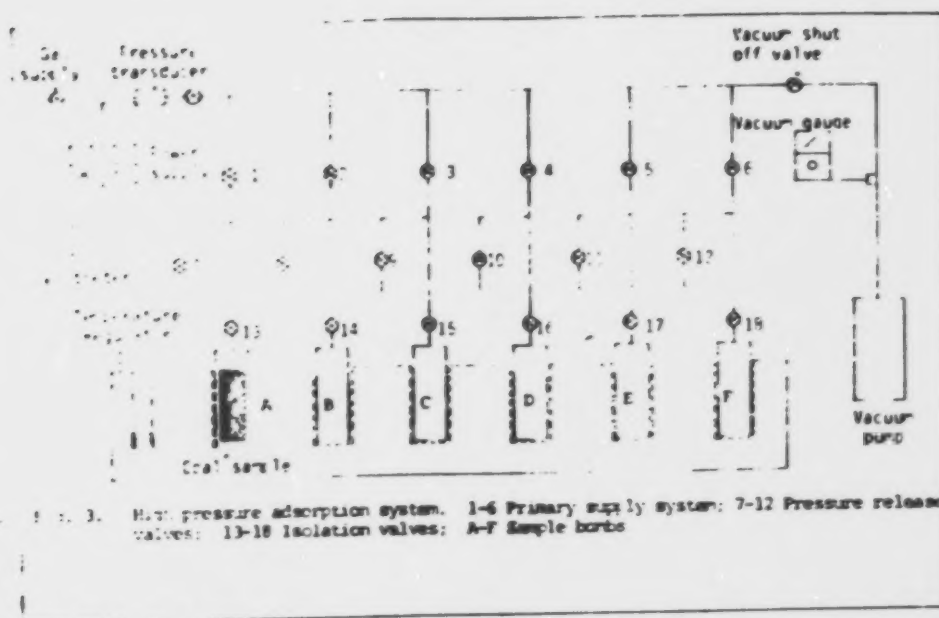
The coal samples are carefully removed from the sample containers and placed into the adsorption bomb. The schematic of the apparatus is given in Fig. 3 and is shown in Fig. 4.

The equipment consists of a gas supply cylinder, which is connected to a manifold through a storage reservoir and a pressure regulating valve. The manifold has six independently controlled outlets to which sample containers are attached using flexible nylon hoses and high pressure quick attach fittings. The manifold is attached to a vacuum pump with a facility to monitor both pressure and vacuum in the manifold, and to the sample containers. Pressure release valves are incorporated to release pressure in the manifold line on any of the sample containers independently.

Sample containers (sample bombs) have isolation valves mounted on them and are placed in a temperature controlled water bath surrounded by a copper sleeve. The copper sleeve permits good thermal conductivity without wetting the bomb surface.

The system is designed so that gas adsorption tests can be conducted up to 7000 kPa gas pressure and at temperatures from -10°C to 40°C . All tests are conducted at the temperature measured in the coal seam.

Before starting tests, sample bombs are calibrated to determine the gas holding capacity of the bombs individually. The bombs are weighed when fully evacuated. They are then pressurised to equilibrium at a defined temperature and re-weighed to determine the gas capacity of the bombs. Calibration curves are obtained for each bomb.



These curves show very clearly that for methane the ideal gas equation is acceptable at pressures up to 4000 kPa, but for carbon dioxide the ideal gas equation is not acceptable at pressures beyond 1100 kPa.

The bomb containing the sample is pressurised to a given pressure. Change in weight is monitored until equilibrium is reached. The increase in weight gives the amount of gas adsorbed in coal plus the amount of gas present in the empty space of the bomb.

Let the weight of the bomb = W_1 (g)

Weight of the sample and bomb = W_2 (g)

Weight of the sample
in the bomb = $W_2 - W_1$ (g)

Weight of bomb and sample when saturated with
gas at pressure ' P_1 ' = W_3 (g)
Density of gas at NTP = ρ_g (cc/g)

Volume of gas (free and adsorbed at NTP)
in the bomb = $\left[\frac{W_3 - W_1}{\rho_g} \right]$

Density of coal (Helium density) = ρ_c (cc/g)

Volume of coal in the bomb = $\left(\frac{W_2 - W_1}{\rho_c} \right)$

Let the volume of bomb (at NTP) = V_0 (cc)

Free volume of bomb not occupied by coal

$$= \left[V_0 - \frac{W_2 - W_1}{\rho_c} \right]$$

Volume of gas (at NTP) contained in bomb at
pressure ' P_1 ' as obtained from calibration
curve = V_1

Free gas volume present in bomb at pressure
' P_1 ' with coal sample contained in it

$$= \frac{V_1}{V_0} \left[V_0 - \frac{W_2 - W_1}{\rho_c} \right]$$

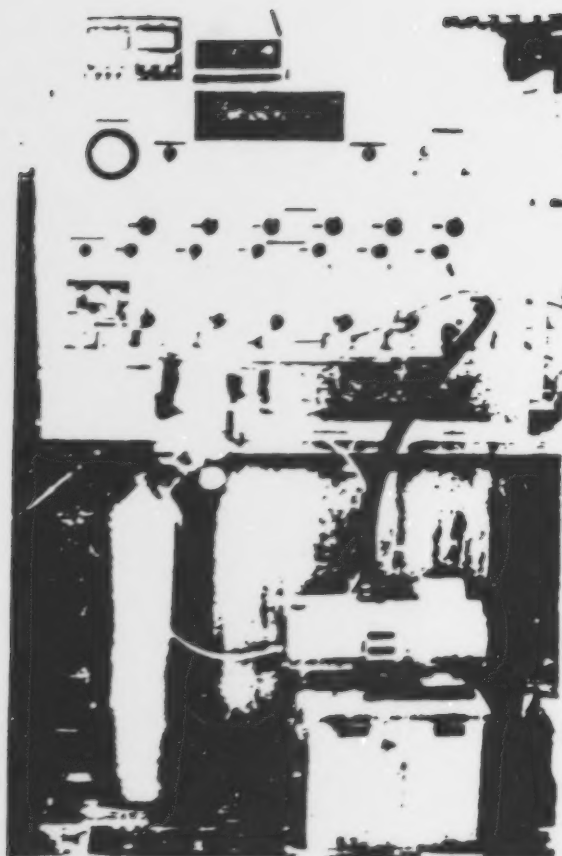


Fig. 1. Apparatus for gas adsorption.

The amount of gas adsorbed in coal,

with

$$\frac{W_2 - W_1}{P_2} = \frac{V_1}{V_2} \left[V_0 - \frac{W_2 - W_1}{P_2} \right]$$

rel.

pre

with

dep

vol.

is

per

des.

The sample is

placed in the

chamber, which

contains the gas

at a pressure and

temperature of the

sample. The

amount of gas

released is

Though the first reading is taken after about 45 seconds, the amount of gas released in the first 45 seconds can be calculated by knowing the total amount of gas adsorbed and subtracting from it the amount of gas present at the time of the first reading. This permits the complete desorption curve for the sample to be obtained.

DESORPTION OF GAS FROM COAL

To calculate the amount of gas present in a coal sample and particularly the lost gas, it is essential to understand the mechanism of adsorption and desorption of gas from coal. A number of studies have been conducted which model gas emission as flow or diffusion. In the absence of knowledge of the various parameters involved such as permeability, diffusion, particle diameter and shape, total quantity of gas present and initial pressure, the use of theoretical formulations is not very useful in calculating desorption of gas from coal under field conditions. Certain empirical formulations, based on simplified relationships, are more useful. These are as follows (Lawa, 1981):

$$Q_t = A \sqrt{t} \quad (3)$$

$$Q_t = A \sqrt{t} \quad (4)$$

$$\frac{Q_t}{Q_\infty} = \frac{12}{d} \sqrt{\frac{D t}{V}} \quad (5)$$

$$\frac{Q_t}{Q_\infty} = \left[1 - e^{-\left(\frac{t}{\tau}\right)^n} \right] \quad (6)$$

$$Q_t = Q \left(\frac{t}{\tau} \right)^n \quad (7)$$

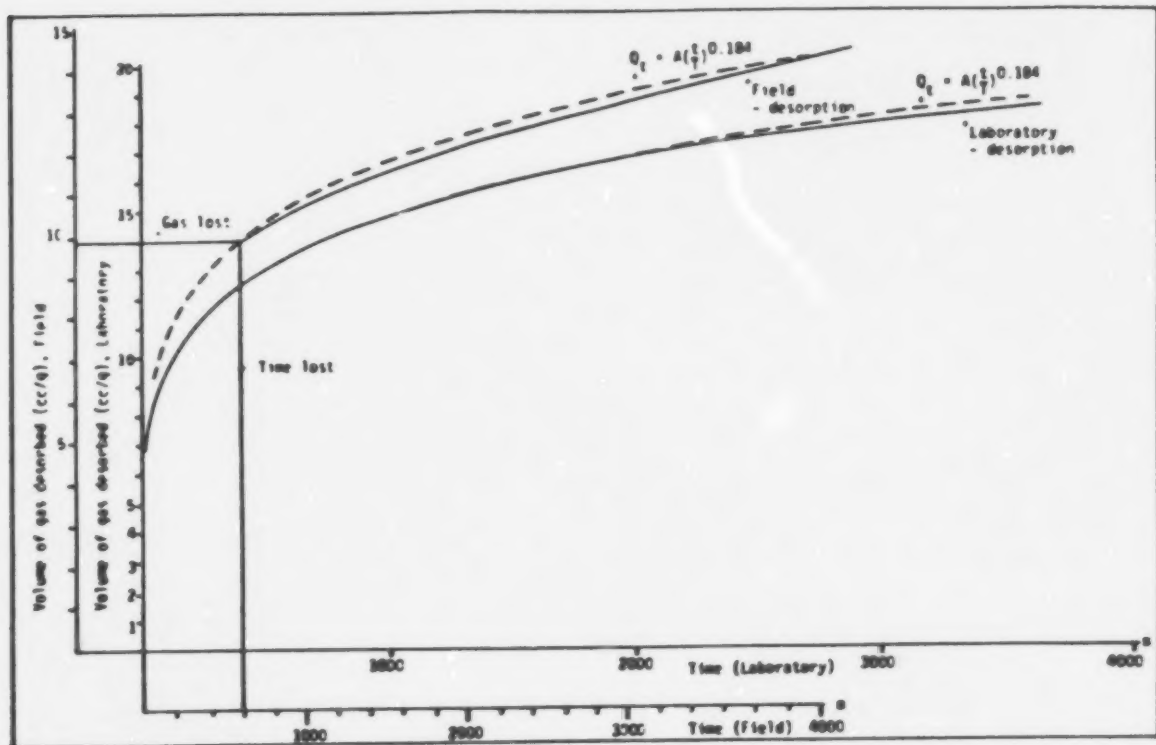


Fig. 5. Results of laboratory and field desorption tests and the use of laboratory determined coefficient n to estimate lost gas. Sample No. 12, gas CO_2 , lost time 600 seconds.

where:

- Q_{∞} = gas desorbed cc/g at time
- Q_t = gas desorbed cc/g in time t ,
- Q = gas present cc/g immediately before desorption,
- A = constant,
- t = time seconds,
- d = equivalent particle diameter,
- D = coefficient of diffusion, cm^2/s
- T = constant

To estimate lost gas it is essential to know the law that a given coal would follow to permit interpolation of the curve.

The method adopted by USPH assumed a square root law (Eq.4). This law was tested under laboratory conditions and it was found that calculating lost gas using this method would be seriously in error.

Table 2 shows the results of calculation of lost gas from the two commonly used equations. These results show that the use of the square root law to calculate lost gas gives consistently low values. It is therefore suggested that instead of the square root law, the modified Eq.7 be used.

Table 2 Comparison of actual and calculated lost gas (laboratory investigation, dry sample, CO₂)

Sample no.	Measured gas lost (cc/g)	Calculated lost gas (cc/g) Eq. 7	Calculated lost gas (cc/g) Eq. 4
1	8.48	8.48	2.71
2	27.72	27.72	0.27
3	14.03	14.02	2.27
4	5.42	5.86	3.26
5	6.90	7.19	4.34
6	6.55	8.05	10.15
7	13.40	13.50	1.84
8	10.51	10.45	1.26
9	6.75	6.56	1.95
10	9.26	9.25	1.61
11	10.50	10.45	0.76
12	6.85	6.84	2.28
13	8.48	8.44	0.29
14	10.40	10.28	1.43
15	13.07	13.07	0.27
16	13.94	13.95	0.30
17	14.14	-	0.10

CALCULATION OF THE VALUE OF THE COEFFICIENT n

The value of the coefficient n (Eq. 7) is calculated by using the desorption data. The results have shown that the value of n to fit the curve is not constant, but depends upon which section of the curve is taken. The values of n as obtained at various test intervals are given in Table 3. The value of coefficient n is variable but it usually decreases with increase in the elapsed time from the start of desorption and the value of T increases. The relationship of (n/T) as a function of elapsed time is given in Fig. 6. It is therefore not possible to assume any value of n , but it has to be determined for the sample under investigation and the elapsed time.

It is possible that the values of n and T are dependent upon pressure. Some initial studies have shown that its value is not very

much affected by the initial pressure within limits (5). The best method would be to determine the gas pressure in situ. In the absence of this data an estimate can be made from the depth parameters. Some data obtained from Australia and overseas countries is given in Table 4. This table indicates that except for special circumstances (close to outcrops, highly faulted and folded areas), pressure could be taken as 0.6 - 0.7 of the pressure at the base of a column of water equal to the depth of cover. This gives a reasonable estimate of the pressure and permits calculation of the coefficients n and T .

CASE STUDY

Coal samples were obtained from surface drill cores. The gas composition was analysed and it consisted mainly of carbon dioxide. The lost time was estimated using the following relationship:

$$\begin{aligned} \text{Lost Time} = & (\text{Time core in sample container} \\ & - \text{Time core on the surface}) \\ & + 0.4 \times \\ & (\text{Time core on surface} - \text{Time core} \\ & \text{extraction started}) \end{aligned}$$

From fixed desorption measurements the amount of gas desorbed cc/g is calculated and is plotted. The best desorption relationship is given by Eq. 7. Therefore,

$$Q_t = A \left(\frac{t}{T} \right)^n$$

The lost gas (Q_b) is desorbed during the lost times (t_o).

$$Q_b = A \left(\frac{t_o}{T} \right)^n$$

(8)

Table 3 Values of coefficients n and T for coal samples, gas CH_4 , Middle Gwynedd seam

Sample no.	Time interval used to compute (s)	Computed value of coefficient (n)	Computed value of coefficient (T)	Coefficient of correlation (r)	$Q \cdot 10^{-5}$
3	61 - 341	0.337	4325	0.997	7.79
	341 - 641	0.256	9821	1.000	2.61
	641 - 941	0.225	13784	1.000	1.65
	941 - 1241	0.249	11118	0.999	2.24
	1241 - 1901	0.223	14581	0.999	1.58
	1901 - 2501	0.206	17041	0.998	1.21
4	340 - 640	0.224	9561	0.996	2.34
	640 - 940	0.211	11407	0.999	1.85
	940 - 1240	0.195	13534	0.995	1.44
	1240 - 1540	0.206	12067	0.995	1.71
	1540 - 1900	0.203	12814	0.994	1.58
	1900 - 2500	0.184	15558	0.999	1.18
5	35 - 335	0.323	3114	0.989	10.37
	335 - 635	0.224	9561	0.995	2.34
	635 - 935	0.205	12705	0.999	1.61
	935 - 1235	0.194	14824	0.999	1.31
	1235 - 1835	0.204	12941	0.999	1.57
	1835 - 2435	0.150	26230	0.997	0.57
6	45 - 345	0.307	4093	0.993	7.5
	345 - 645	0.228	10294	0.998	2.22
	645 - 945	0.232	10023	0.999	2.31
	945 - 1245	0.208	12877	0.999	1.62
	1245 - 1845	0.203	13750	0.990	1.68
	1845 - 2445	0.210	12668	0.997	1.45
	2445 - 3045	0.210	13817	0.999	1.32

Table 4 Gas pressure in relation to depth in Australian and overseas coal mines

Mine (coal seam)	Gas	Depth (m)	Gas pressure (kPa)	Gas pressure hydrostatic head
AUSTRALIA				
West Cliff (Bullis Seam)	CH_4	680	3800	0.68
Apple (Bullis Seam)	CH_4	130	4200	0.77
Leichhardt (Bullis Seam)	CH_4	680	3640	0.66
Collinsville (Bullis Seam)	CO_2	280	1580	0.54
Metropolitan (Bullis Seam)	CO_2	610	3900	0.69
U.S.A.				
Grandstone 1487 (Coal Bed)	CH_4	600	4120	0.72
Pittsburgh Coal Seam	CH_4	252	1280	0.54
Metropolitan (Bullis Seam)	CH_4	470	4880	0.95

Note: Hydrostatic head is taken as equal to depth without assuming the depth of water table.

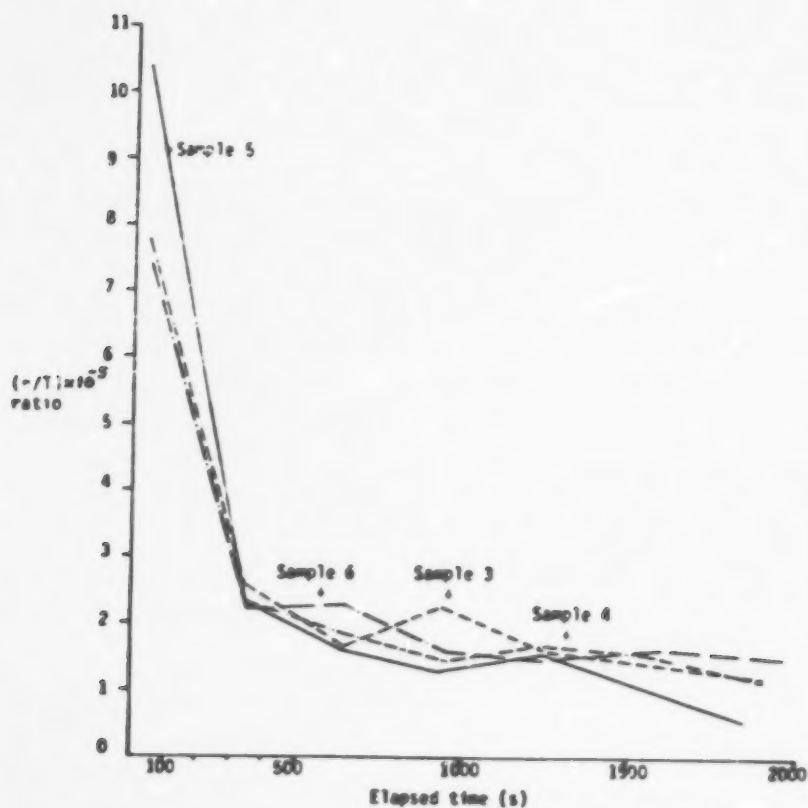


Fig. 6. Values of coefficient (n/T) as a function of elapsed time.

Table 5 Calculation of lost gas using the Eq. $Q_t = A(t/T)^n$
 $\left\{ \begin{array}{l} t_0 = 900 \\ n = 0.293 \end{array} \right\}$

Desorption TIME s	VOLUME DESORBED	$\ln(Q_t)$ Y	$(t_0)^n$	$(t+t_0)^n - t_0^n$	$\ln \left[\frac{(t+t_0)^n - t_0^n}{X} \right]$
60	0.085	-2.464	7.338	0.140	-1.966
120	0.181	-1.711		0.274	-1.294
180	0.250	-1.387		0.403	-0.910
240	0.319	-1.143		0.526	-0.642
300	0.420	-0.867		0.645	-0.438
360	0.497	-0.699		0.760	-0.274
420	0.572	-0.559		0.871	-0.138
480	0.638	-0.449		0.979	-0.021

Therefore the real amount of gas desorbed

$$Q_t + Q_o = A \left(\frac{t+t_o}{T} \right)^n$$

Taking logs

$$\ln(Q_t) = \ln\left(\frac{A}{T^n}\right) + \ln[(t+t_o)^n - t_o^n]$$

A plot of $\ln(Q_t)$ against $\ln[(t+t_o)^n - t_o^n]$

should be a straight line relationship with slope approaching 45° . The intercept gives the value of $\ln\left(\frac{A}{T^n}\right)$.

The amount of gas lost (Q_o) can be calculated by introducing the value of t_o into Eq. 8. The total gas desorbed is then calculated by adding the gas lost to the gas desorbed from actual measurements. Table 5 gives details of calculations. Fig. 5 gives results of actual measurements and the desorption tests conducted on field samples. The close approximation of the curve obtained from calculated values of n based on laboratory tests for the assumed time to those based on field investigations is acceptable. The field and calculated desorption curves in the region of lost time lie very close to each other.

CONCLUSIONS

The gravimetric method of measuring gas desorbed from coal samples, both in the field and in the laboratory, is superior to the volumetric method commonly used. It has also been shown that the square root law of estimation of the gas lost during field estimations does not hold good, as determined under laboratory conditions. The most suitable empirical relation is given by

Eq. 7. The coefficients n and T are dependent upon the time elapsed from start of desorption. These can be determined reasonably accurately by repressurising the sample and measuring the gas desorbed over a short period of estimated time lost.

REFERENCES

- Airey, E.M. (1968). Gas emission from broken coal. An experimental and theoretical investigation. *Int. J. Rock Mech. Min. Sci.*, Vol. 5, pp. 475-494.
- Belin, J. (1971). Measurements of the gas concentration in coal seams - direct method. *Maitrise du degagement grisouteux, Luxembourg, Commission of the European Communities, EUR 4670 df*, pp. 39-56.
- Bertard, C., Bruyet, B., Gunther, J. (1970). Determination of desorbable gas concentration of coal (direct method). *Int. J. Rock Mech. Min. Sci.*, Vol. 7, pp. 43-65.
- Diamond, W.P., Levine, J.R. (1981). Direct method of determination of gas content of coal: Procedures and Results. *USM, Rep. Invest. R.I. 8515*, 36 p.
- Ettinger, I.I., Lidin, G.D. Dimitriev, A.M., Shapachina, E.S. (1958). Systematic handbook for the determination of the methane content of coal seams from the seam gas pressure and the methane capacity of the coal. *Institute Mining, Academy of Sciences, USSR, USM Translation No. 1505* 29 p.
- Graham, J.I. (1937-8). The measurement of quantity and pressure of methane in coal. *Trans. Inst. Min. Eng.*, Vol. 94, No. 122.

Kin, A.G. (1977). Estimating methane content of bituminous coal beds from adsorption data. USBM, Rep. Invest. No. 8245, 22 p.

Kissel, F.N.; McCulloch, C.M.; Elder, C.H. (1973). The direct method of determining methane content of coal beds for ventilation design. USBM, Rep. Invest. RI 7767, 22 p.

Lana, R. (1980). Drainage of gas from the solid at West Cliff Colliery: Optimisation of drainage hole design parameters. GCM Report No. 18, Division of Applied Geomechanics, C.S.I.R.O., 76 p.

Lana, R. (1981). The use of adsorption-desorption isotherms in predicting outburst conditions, GCM Report No. 21, Division of Applied Geomechanics, C.S.I.R.O., 76 p.

Lidin, G.D.; Petrovyan, A.E.; Dimitriev, A.M. (1965). Determination of gassiness of coal seams and methods for the prediction of methane emission in coal mines of the USSR. Int. Conf. Directors of Mines Safety Research, Sheffield, U.K. SMRE Translation No. 5158.

Moak, K. (1976). Effect of face length, rate of advance and type of stowing on methane emission. Gluckauf, Vol. 112 (20), pp. 1156-1159.

Moak, K.; Treskov, A.V.; Paul, K.; Mucke, G.; Duerpe, G.; Koppe, W. (1977). Control of gas emission. Forschungshafte Kohle, Luxembourg, Commission of the European Communities, EUR 5582d, p. 145.

Tarnowski, J. (1960). A simplification of a method of determining gas pressure by means of a core barrel (Kriczewski's design). GIG, km. No. 243.

Winter, K.; Tanas, H. (1975). Describable methane content and gas emission parameters of coal. Gluckauf-Forschungshafte, Vol. 36 (3), pp. 114-119.

Tarnowski, J. (1972). Possibility of assessment of methane content of coal in a seam from samples taken from ribside. GIG, km No. 570, 5 p.

DISCUSSION

R.J. WILLIAMS (Collinsville Coal Company Ltd.): It is suggested that the lost gas calculations or the lost gas determination from the laboratory method described may be substantially over what really is the case. It was indicated that the zero time is at the point when the coal is cut at the bottom of the hole and under those conditions there is a hydrostatic head of water on top of the coal. As well the core being cut is confined ultimately by the triple tube core barrel so there is the point of deciding when to make that zero time. It could be that only when it is out of the triple tube core barrel, the desorption rate is similar to what has been done in the laboratory. Up until that time

there is resistance to flow due to confinement and to the core barrel. Up until that time a much lower desorption rate occurs so it could be that the true lost gas value is somewhere between what has been calculated and what the U.S.B.M. shows.

B. LAMA (Kembla Coal and Coke Pty. Ltd.): What has been shown basically is the technique. The question of time, when does the first desorption start will depend upon how the drilling is done and what the local conditions are. Yes it is true, the time is estimated. For example, in the paper, the lost time has been estimated from the time pulling out that core is started to the

time it reaches the surface but multiplied by the factor of 0.4. That is the lost time which has been taken as if the hole is full of water. It is assumed that the gas pressure is only about 0.6 of the vertical depth. It is not assumed that the gas starts liberating from the sample immediately on starting to pull the sample out, but it starts liberating only if it has reached a certain amount of depth where the gas pressure in the coal sample is equal to the hydrostatic head of water in that particular hole. The figure shown had two scales, one laboratory and one field. The reason behind that was that knowing very well that pressure might influence, the samples were subjected to higher pressure in the laboratory. Using the data even from very high pressures and correlating it with the field data, it was still found that the correlation is pretty good. The factor n is not influenced by the pressure. The factor obtained in the laboratory could be used to transfer the data to the field conditions although the laboratory pressure conditions are quite different than the field conditions.

R. KING (U.S. Bureau of Mines): Concerning the dry sample which was used. Could not the over-estimation of the laboratory versus the lost gas be due to the fact that the sample was dry? The same problem is experienced with isotherms. With a dried isotherm sample the theoretical maximum gas storage is much higher than what it is in reality because the moisture and the gas are both trying to share the same hole. Has this factor been addressed in this over-estimation?

R. LAMA: Yes, the effect of moisture on the adsorption of gas in the sample is very large. The figures vary from anything from 10-15 per cent. The question arises is, that the sample would not dry if the sample is subjected as

early as possible to the laboratory conditions. It is not necessary to evacuate the sample. Normal absorption tests are done where all the gas is first evacuated and then gas is pumped in it. The field samples are not evacuated. Neither are the samples evacuated in the laboratory tests. They are kept as they are, and that is true, though a certain amount of water still gets out from the coal sample. But the question is that it is not an attempt to determine the amount of gas which is present in the coal seam from indirect method using sorption, it is an attempt to determine the value of the parameter n . Results show that higher pressure adsorption puts in more gas (with or without water). The amount of gas though put into the sample is larger, but the value of n does not change. German investigations using this formulation do show that even the value of n is more or less independent of the grain size of the particles. Therefore it is believed that though this is a bit more time consuming technique, the limitation of the amount of gas lost is far better. The determination of gas lost is far more accurate.

I. GRAY (Australian Coal Industry Research Laboratories): Actually there seems to be a varying boundary condition as the sample is raised. The sample is cored and then raised up the borehole and the hydrostatic head over the sample is changing entirely during that period. If the empirical 0.4 is correct then that is fine, but it must be empirically done. The second thing is that in working in combination with Collinsville Coal Company, A.C.I.R.L. found that comparisons of field tests at Collinsville Coal using their desorption and back analysis procedure gave a very close correspondence indeed, which gives some faith in the method. It is a log-time back analysis, taking into account when desorption would commence and field values measured underground using pressure and sorption curves. If it is to be researched further,

possibly that means going into the realms of simulators and multi phase flow, but possibly it is not worth it if a reasonable comparison can be obtained which appears to be the case with this particular mine.

R. LAMA: The comments made were not on use of some sort of quadratic equations to fit desorption of gas from the coal sample as used by Collinsville. The comments were made on the standard methods which have been accepted world wide for estimating the lost gas. It is possible that the quadratic equation can be fitted in to the data and pretty close values may be obtained. After all that is a question of curve fitting. The aim was to design the system so that with values of coefficients determined under similar conditions on the same coal sample, an interpolation could then be made.

That seems more scientific rather than trying to fit a quadratic equation. I do say, no attempt was made to use any quadratic equations. The number of constants that would be involved and have to be determined would be so many. It seems better to use a simpler formulation.

W. NIGHTON (National Coal Board, U.K.): Looking mainly to the benefits that have to be obtained from this research work it must be agreed that it is worthless doing research if there is not some production benefit. The requirement was mentioned for a 50% capture by methane drainage to maintain acceptable levels. Does this mean for the working seam or for the total gas emission from the strata above and below the working seam?

N.C.B. experience has been that only 3 to 5% of the gas from the working seam is captured by a cross measure methane drainage system. Tracer gases were used to conduct these trials. Finally, apart from using the prediction work

in headings, what is the real value of determining the total gas emission from longwall districts. N.C.B. experience up to now has been that there has been an error of as much as 2 to 1 in the predicted figures. It was found that predictions are more reliable when based on past experience.

R. LAMA: That is a very interesting question. In referring to the amount of coal which can be produced from a longwall face, only the gas contained in the coal seam, not of the surrounding strata was taken into consideration. Also the assumption in the paper was based on some of the observations from driving of headings, that possibly 30-40% of gas is liberated in the district. Maybe up to 50%. The assumption was made that only 50% of the gas contained in the coal seam would be liberated at the return end of the longwall face. It is agreed, it is very hard to determine the total amount of gas that would be liberated in a district, whether it is longwall mining or bord and pillar development. The experience with some calculations for West-cliff Colliery, was that they were wrong between 100 and 150%. The thing which will help to understand will be having such theoretical models, or using different types of formulation for different types of models and identifying which model possibly gives results which are closer to what are observed in practice. The calculations cannot be done purely on any mathematical or any hypothetical model without confronting those calculations with actual measurements. The difficulty arises when there are no data, particularly in some of the cases where there is no experience of what will happen. In such cases there is no other option except to go to some of those models. In the paper by Marshall, Lama and Tomlinson (to follow) some data will be presented in which some 12 different methods are used. These methods have been used overseas to calculate the amount of gas that will be liber-

ated in a longwall operation. It is found that values could be anything from x to $5x$. It is very good for people in Europe, having had an experience. West Cliff has used the method that people use in Germany, the U.K., France, Poland, Russia and Belgium. Using these methods and

adapting them to West Cliff conditions, the amount of gas emissions arrived at are so large that these are sometimes incomprehensible. Once these methods are used and confronted with actual measurements, probably a better answer to this problem of gas emission will be obtained.

AR-124

APPENDIX B

FACTORS AFFECTING GAS RELEASE
FROM THE WORKING SEAM

HAYES, P.J.; 1982

FACTORS AFFECTING GAS RELEASE FROM THE WORKING SEAM

By

P. J. HAYES¹ABSTRACT

The seam gas content at Westcliff Colliery is 12-15 m³/tonne of coal. Composition ranges from 99% CH₄ to 52% CO₂ in one area. Desorption rates are proportional to gas contents and CH₄ is preferentially desorbed. A crushed zone of reduced permeability advances with the face - at Westcliff ribside crushed zones are 2.5 to 6.5 m wide. Poor strata control widens crushed zones. 12 months of pre-drainage allowed 2 heading development and longer pillars to be used in a virgin area instead of 3 or more headings without drainage. Machined coal may lose as much as 1.6 m³/tonne being conveyed out of Westcliff Colliery. Experience differs from mine to mine. Permeation is affected by time, strata movements, changing water content, and changing stress. Permeabilities vary from 10 md for Pittsburgh Seam to less than 1 md for the Bulli Seam. All these data and differences await further investigation.

INTRODUCTION

The emission of firedamp from coal into mine workings is a complex combination of processes. Seam gases flow into mine workings only as a result of the pressure drop between the gas within the coal seam and the atmospheric pressure in workings. As a face advances, the gas pressure in the virgin coal falls and the adsorption equilibrium is disturbed, resulting in the release of some previously adsorbed gas into the microfractures and cleats in the face area.

¹Graduate Mining Engineer, ALS Appin Colliery formerly at West Cliff Colliery.

Gas release continues from outbye ribs and from coal already won. The rate and extent of gas release from the working seam depends mainly upon: 1. gas content, 2. gas composition, 3. position of the abutment zone, 4. the degree of breakage of cut coal, and 5. changes in permeability.

GAS CONTENT

The gas content of coal, volume per unit mass, m³/tonne or cm³/g, depends primarily on seam gas pressure, strata temperature, adsorptive capacity and moisture content, and to a lesser extent on permeability and distance from outcrop. The content steadily decreases from the time coal exists in a virgin state to the time of its utilisation. Desorption of gas from coal is proportional to the gas content, providing all other factors remain constant, as shown by the cumulative desorption curve for a Bulli seam sample, Fig 1. If the gas content halves, the gas emission rate should also halve. Pre-drainage at West Cliff Colliery NSW, of 470 panel by drilling boreholes from an adjacent panel reduced the gas content of 470 panel coal from a virgin value of 12-15 m³/tonne to less than 4 m³/tonne in seven months. Within 12 months gas content had reduced to less than 1 m³/tonne, with the result that no gas problems were experienced in driving the two-heading development through what was otherwise virgin ground.

GAS COMPOSITION

The 2 principal components of most seam gases are carbon dioxide and methane. Composition can vary from nearly 100% CH₄ to nearly 100% CO₂. Minor constituents include nitrogen, ethane, helium, hydrogen and higher hydrocarbons.

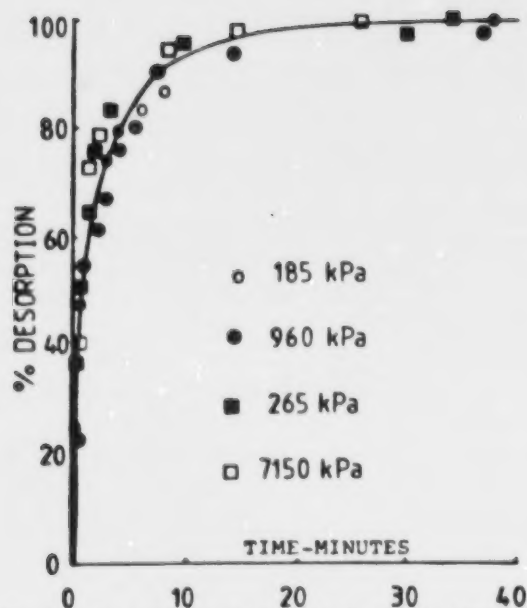


Fig.1 - Desorption of sample E, Bulli Seam, West Cliff Colliery (-53 μ m) at 30°C for a range of equilibrium methane adsorption pressures.

For the same seam, large composition variations can occur over relatively short distances. At West Cliff Colliery the CO_2 concentration in the Bulli seam gas, as sampled from in-seam holes, varies from less than 1% CO_2 in the Southern workings to over 50% in the Northern workings, as illustrated by Fig.2. Coal has a higher adsorptive capacity for CO_2 than CH_4 . Fig.3 illustrates this by a plot of the dry, 30°C equilibrium desorption isotherms of CH_4 and CO_2 for Bulli seam coal from West Cliff Colliery. The implication is that for any given seam pressure, the greater the proportion of CO_2 in the seam gas, the higher is the combined overall emission rate providing the seam pressure remains the same. At the same adsorbed gas concentration, CH_4 desorbs more rapidly than CO_2 , so that for a coal seam district containing both gases, the adsorbed CO_2/CH_4 ratio increases with time. In other words, the Concentration of seam gases as measured from samples from in-seam holes may

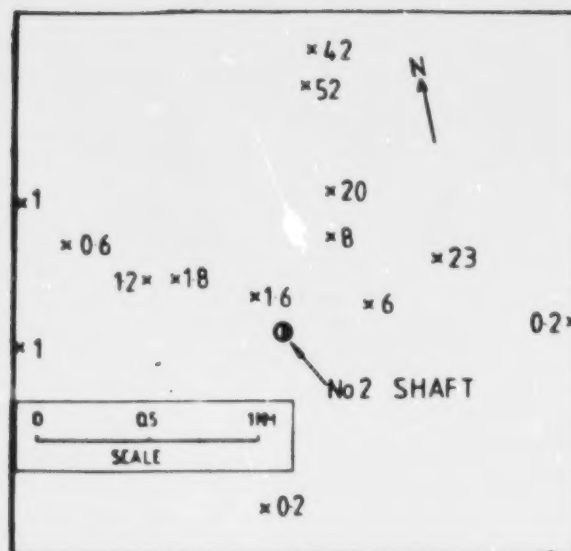


Fig.2. - CO_2 concentration in Bulli Seam gas at West Cliff Colliery, N.S.W.
e.g. x 1 = 1% CO_2

not be exactly the same as the original adsorbed gas mixture.

ABUTMENT INFLUENCES

The solid coal ahead of the working face can be divided into two sections, one section on the face side of the front abutment and the other beyond the abutment into the solid coal. The region on the face side of the front abutment is called the fractured or crushed zone. Kissell(1972) detailed evidence for the existence of this zone, including its increased permeability compared to the zone beyond the abutment into the solid coal. Due to fracturing within the crushed zone, the permeability is much higher than in the intact zone beyond, which exhibits virgin permeability. As the previous crushed zone is mined, new crushed zone is formed ahead of the face. Calculations of the extent of the crushed zone at West Cliff is shown in Fig. 4.

Fully caved longwall faces exhibit a feat-

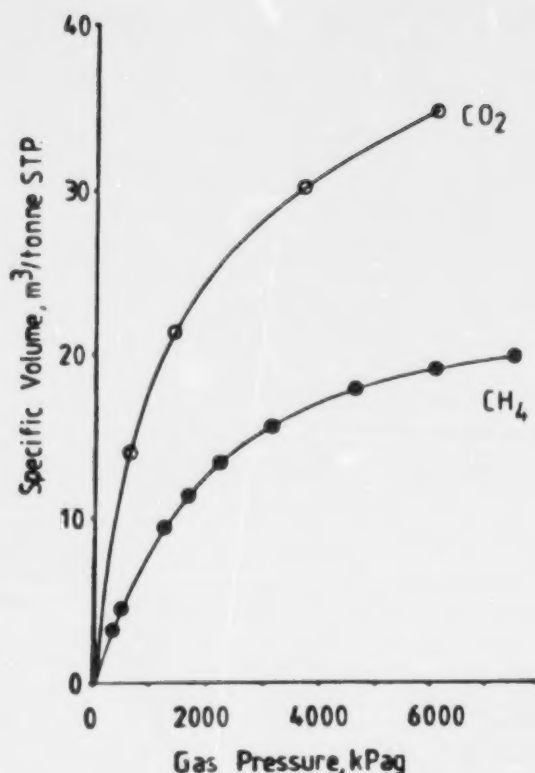


Fig. 3. Equilibrium Desorption Isotherms for Bulli Seam coal, West Cliff Colliery, 30°C, 0% moisture. (Hayes, 1982).

ure which Ettlinger (1958) called the "gas barrier". The gas barrier is, in fact, the front abutment line, beyond which little gas flows from the coal seam into the workings. Most of the gas from the working seam on longwall faces originates from the fracture zone, which is in most cases 5 - 10 m ahead of the face. The existence of a gas barrier can be justified in terms of the effect of stress on the permeability of coal. Somerton (1975) found that increased applied stress in constrained specimens of coal caused a decrease in permeability of several orders of magnitude up to the point where microfracturing occurs, beyond which permeability increased. This indicates that the abutment stress (which is greater in magnitude than the virgin vertical

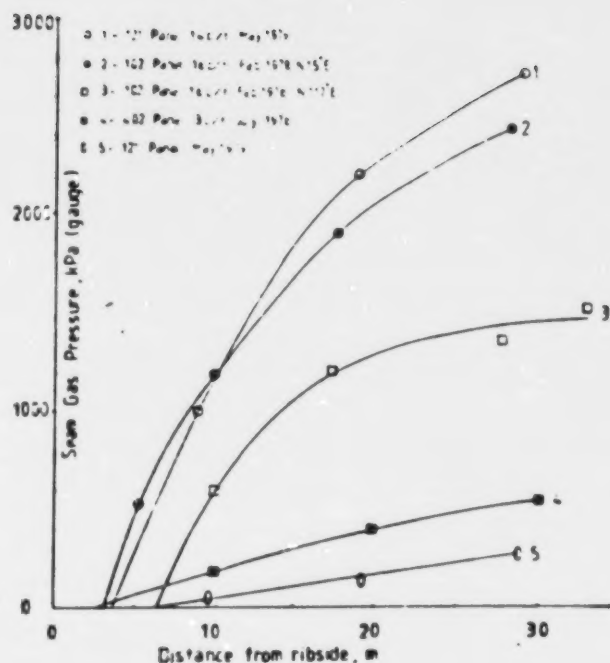


Fig. 4. Variation of seam gas pressure with distance into solid coal, West Cliff Colliery, Appin. The depth of the crushed zone can be calculated where the pressure is zero, and varies from 2.5 to 6.5 m.

stress) could decrease permeability to the extent that the region of high vertical stress concentration acts as a gas barrier, discouraging the flow of gas from beyond the abutment zone into workings. Jolliffe (1970) stated that the work done by British M.R.E. found that gas migrated only about 15 m from solid ribs in longwall gates adjacent to goaf areas, and this accords with the concept of a gas barrier. On faces which achieve poor strata control, with excessive abutment loads leading to extensive fracturing ahead of the face, gas emission from the seam into the working area is increased due to the widened crushed zone: poor roof control can lead to increased gas emission.

Gas pressure measurements made in subadjacent seams show that the gas pressure profile

is similar in shape to the ground stress profile in the vicinity of longwall workings (Mucke, 1975) and depends on time, roof support method, method of working and face advance rate. The profile of flow rate is also related, although inversely, to the ground stress beyond the abutment zone. Permeability is the controlling factor on the face side of the abutment. The flow profile along a borehole at West Cliff Colliery intersecting a shear zone is shown in Fig. 5. The abutment and crushed zones are clearly evident, as well as the distressed nature of the shear zone.

In a study of the Barnsley seam at Yorkshire Main pit, Richards (1978) found that for one advancing face practising methane drainage, 74% of the gas in the air at the outbye end of the district return originated between the two 10 m statutory intake and return sampling points. Of the gas between the two points, 48% came from the face, roof and floor and 52% from the goaf at the tailgate. Of the gas which originated from the seam, 66% emanated from the major cleats in the face 1.5 m apart. These results implied that about 35% of the total gas made from the district originated from the exposed face of this advancing longwall. Pre-drainage of the seam, if possible, would have resulted in a major contribution to methane control (Richards, 1978, p. 702). Pre-drainage of a retreat longwall block should then significantly reduce gas problems on the face itself, as well as, more importantly, reducing emission into the next developing gate roads to the point where it does not retard advance rates. The usefulness of pre-drainage was illustrated by the driving of a two-heading development through a virgin district at West Cliff Colliery, previously pre-drained for up to 12 months. Without pre-drainage, a minimum of three headings (two outside returns) would have been required, and pillar lengths would have been reduced.

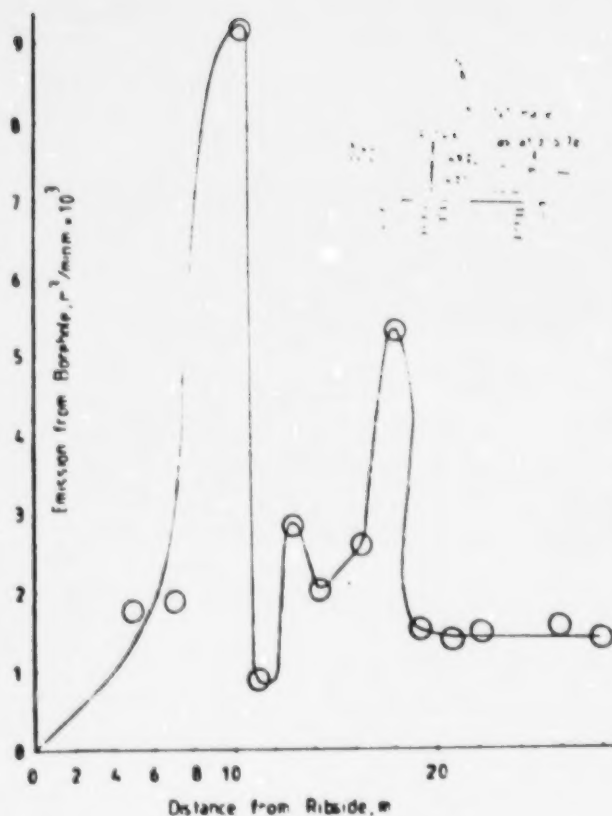


Fig. 5. Flow profile of a borehole in a virgin area of West Cliff Colliery, in the vicinity of a small shear (outburst) zone. Abutment: 10-12 m
Shear zone: 18 m

EMISSION FROM BROKEN COAL

Coal which has been cut and transported out of the mine, or coal spalled from ribs in mine roadways, can contain appreciable quantities of gas in the desorbed state. The Pittsburgh seam, U.S.A., for example, has been recorded to contain up to 80% of its original gas content in lump coal collected at the faces of development workings (Kissell and Deul, 1974).

Possibly the earliest work on residual gas content in mined coal was carried out by Graham (1921). From previous work which indicated that seam gas desorbed very slowly even from small lumps (25 mm), Graham concluded that cut

coal retains almost all of its original gas for a relatively long time and that residual gas contents are a good index of gassiness. Hinsley's work on samples of face coal (Hinsley, Konda and Morris, 1965) indicated that "unless the coal is permeable and there is much fire-damp emission from the roof and floor, the residual firedamp content of the coal should give a rough guide to the likely emission that will occur when the seam is worked". However, no correlation was found between methane emission at the statutory 10 m points and residual methane content and, in contrast to Graham's earlier work, Hinsley did not conclude that mined coal retained most of its original gas content. Later work at the Pittsburgh Research Centre by Kissell and Deul (1974) on the subject of coal breakage and methane emission found that the Pocahontas No. 3 Seam contains less than 10% of its original gas content when sampled from development faces. In this seam, then, 90% of the original gas content has escaped before the coal becomes the working face. If only 10% of the original gas remains on the face, then the size to which it is broken can make little difference to mine gas make. Some coals, however, retain up to 80% of their original gas content when exposed as face coal (Kissell and Deul, 1974, p. 183). Nevertheless, coal breakage still accounts for only a minor percentage of total gas emission. This was shown in studies of the Pittsburgh Seam (Centinbas, 1972) which concluded that breakage accounts for 10-15% of the total gas emission. Pickering (1969) found that gas from coal breakage on a typical British long-wall face represented only 8-15% of the gas make between the two statutory 10 m points.

Although complete data are only available for several American and British Seams, it seems reasonable to assume that breakage is a minor contributor to methane emission in other seams as well. In seams like the Pocahont-

as No. 3, gas release occurs over a relatively short time and little gas remains in face coal to be released during breakage. In seams like the Pittsburgh, gas release occurs over a relatively long time and insufficient time is available from exposure on the face until it reaches the surface to pollute the mine atmosphere with firedamp. With reference to the Bulli Seam, there are several indirect indicators as to the contribution of cut coal to the total methane output of the mine. Firstly Hargraves' work at Metropolitan Colliery (1963) on desorption of gas from coal cuttings from boreholes drilled ahead of the face indicated a residual gas content of not much more than $1.0 \text{ m}^3/\text{tonne}$. Work done at West Cliff Colliery by Griffiths, cited by Lama (1980), showed for one panel that the maximum gas released by breakage and rib emission within the immediate face area was less than $1.0 \text{ m}^3/\text{tonne}$. In one face sample taken from a rapidly advancing development heading at West Cliff, the total gas content (composition 99% CH_4) was calculated as $10.8 \text{ m}^3/\text{tonne STP}$. From the desorption curve of the sample shown as Fig. 6, it can be seen that for the minute or so that the coal would be in the face area after cutting, only 4% of the desorbable gas content would be emitted, and that in the 30 min required to reach the surface, the coal would have emitted less than 15% (i.e., $1.6 \text{ m}^3/\text{tonne}$) of the desorbable content. Since the overall mine output of methane is about $40 \text{ m}^3/\text{tonne}$ (not including drained methane), it is obvious that the major source of gas at West Cliff is the coal ribs, and that the residual gas content of face coal only contributes in a minor way to the total mine gas make. However, even though the gas in face coal contributes little to total mine gas make, it can be a significant factor in pollution of the face area. Emission from cut coal could increase general body gas at the face by 0.1-1.0% CH_4 , depending upon residual gas content

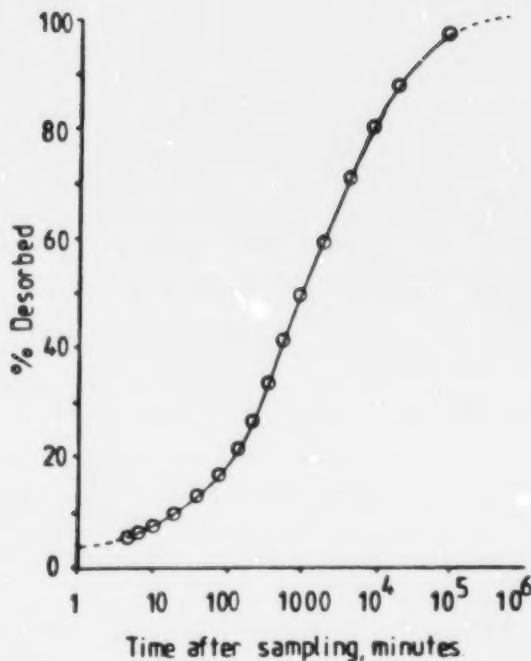


Fig. 6. Desorption of a face sample from West Cliff Colliery: -73 ± 3 mm, original gas content $10.8 \text{ cm}^3/\text{g}$; 0.9% moisture; seam gas 99% CH_4 .

(1-11 m^3/tonne), air quantity (2.4 - 9 m^3/s) and loading rate (4-8 tonne/min).

PERMEABILITY EFFECTS

In-seam boreholes have enabled much basic information about the relationship between coal and the contained seam gas to be obtained. This information is of great help in designing degasification layouts and other methane control techniques. Boreholes were used in-seam at West Cliff Colliery to determine background information on the nature of flows and pressures in the Bulli Seam (Hall, 1978) and to attempt to optimise borehole spacing for in-seam degasification (Lama, Griffiths and Marshall, 1980). Most information obtained from boreholes relates to the permeability or fluid conductivity (Mordecai and Morris, 1974) of the coal seam. Permeability refers to the ability of the coal to transmit gas when a pressure or concentration gradient exists across it.

One major characteristic of coal seam permeability is that the measured permeability to methane appears to increase with time. This was first reported by Kissell (1973), who found that solid ribs of older workings of a mine were more permeable than ribs of new workings. Zabetakis surveyed the rib emissions of a gassy Pittsburgh seam mine (1972) and found that sections of rib close to the working face and freshly exposed by mining were emitting gas at the same rate as comparable rib hundreds of metres outbye and exposed for many months. Preliminary work at West Cliff Colliery indicated a similar effect. If the rib permeability to methane was constant, this could not have occurred since older ribs would yield much less gas than freshly exposed ribs near the face. Several factors could account for this increase in permeability. One is the destressing or relaxation that results from strata movements that accompany mining (Stewart, 1971). Another is the coal shrinkage resulting from the loss of methane (Hoffat and Weale, 1955). Kissell (1975) reasons that the most probable explanation is a relative permeability effect in which the flow of methane is partly controlled by the degree of water saturation in the coal seam: the permeability to methane increases as the water in the coal decreases and allows more pore space available to the gas phase. This assumption was based on the observation that as the waterflow from in-seam holes decreased, the gas flow increased to a peak, then slowly dropped off to a lower flow rate. Stewart (1971), discussing work done in the Bulli seam at Metropolitan Colliery found a similar increase in gas flow from a borehole in a freshly exposed conventional face and comments that the flow pattern is more consistent with an increased average diffusivity than with the opening up of shrinkage fissures. Increased diffusivity could have been the result of reducing the water saturation of the coal. Barron (1978), using an air-injection technique determined that the permeability of

the central portion of a 40 m wide pillar increased remarkably between the time of formation and six months later. He subscribed the phenomenon to a reduction in the strength of the pillar caused by stress effects, but he admitted that no obvious physical deterioration had occurred in the structural integrity of the pillar. Perhaps a more plausible explanation of the reduction in permeability of the central portion of the pillar would be that it had in fact 'dried out' and the moisture loss had occasioned the increase in permeability.

The nature of the coal seam appears to have as equally important effect as overburden pressure or stress effects on permeability. Results obtained for permeabilities of U.S. and European coals have indicated that the U.S. seams are generally more permeable and therefore more amenable to in-seam degasification (Curl, 1978). The in-situ permeability of the Pittsburgh seam has been measured at about 10 millidarcies (md) (Kissell, McCulloch and Elder, 1972), whereas the Bulli Seam permeability is considerably less than 1 md (Lama, 1980). Fisecki (1975), reworking some of Margraves' data for the Bulli Seam (Stewart, 1971), calculated Bulli seam permeability as less than 10^{-3} md compared with values in excess of 2 md for Canadian coals studied. Free flow rates (i.e. unaided by vacuum) of gas flow from Pittsburgh in-seam holes averaged 20 m³/day/m and Pocahontas No. 3 holes averaged 5 m³/day/m, whereas Bulli Seam holes at West Cliff Colliery have averaged about 3 m³/day/m, illustrating that gas flow rates are dependent upon many variables, including permeability. Permeability in a seam may also be directional, depending upon the position of major and minor cleats, permeability being greatest in the direction of the surface of the major cleat. Flows from holes drilled parallel to the major (face) cleat at West Cliff Colliery were found to be lower and sustained flow for a shorter

time than holes drilled perpendicular to the major cleat (Hall, 1978).

CONCLUSION

From the foregoing, it is obvious that the nature of gas emission from the working seam into mine workings is a very complex one, resulting from the interaction of a variety of different factors, including gas content and composition, stress effects and permeability changes, as well as degree of breakage at the face. Further investigation of these factors and application of the knowledge obtained to such processes as gas drainage can only lead to a reduction in problems of gas accumulation and outbursts.

ACKNOWLEDGEMENT

The author thanks Kembla Coal and Coke for its support of the work and permission to publish the paper.

REFERENCES

- Barron, K., 1978. An air-injection technique for investigating the integrity of pillars and ribs in coal mines, *Internat. J. Rock Mech. and Min. Sci.*, v. 15, p. 69.
- Centinbas, A. et al., 1972. Methane and dust control by water infusion - Pittsburgh coalbed, U.S.B.M. Rep. Inv., 7640, 17 pp.
- Curl, S.J., 1978. Methane prediction in coal mines, *Internat. Energy Agency Coal Research Report No. ICTIS-TR 04*.
- Ettinger, I.L., 1958. Systematic handbook for the determination of methane content of coal seams from the seam pressure of the gas and the methane capacity of the coal, *Inst. Min. Acad. Sci. U.S.S.R.*, 29 pp. M.C.B. Translation A1606 SEH.
- Fisecki, M.Y., 1975. Methane pressure and flow measurements in coal and surrounding strata, *C.I.M. Bull.*, Oct., v.68, No. 762, p. 25.
- Graham, J.I., 1921. The adsorption or solution of methane and other gases in coal. *Trans.*

- I.M.E., v. 62, p. 298.
- Hall, R.T., 1978. Progress report of gas emission studies for West Cliff Colliery, A.C.I.R.L. Report 301A, May.
- Hargraves, A.J., 1963. Gas in face coal, Trans. Aus. I.M.M. March, p. 7.
- Hayes, P.J., 1982. Some aspects of gas emission from the Bulli Seam, M.E. thesis, Univ. of N.S.W. (in preparation).
- Hinsley, F.B., Konda, B., and Morris, L.H. 1965. Estimation of the firedamp content of coal samples, The Min. Engr., July, p.591.
- Jolliffe, G.V., 1970. The emission of methane from rapidly advancing faces, The Min. Engr., Feb., p. 325.
- Kissell, F.N. and Deul, M., 1974. Effect of coal breakage on methane emission. Trans. A.I.M.E., v. 256, June, p. 182.
- Kissell, F.N., McCulloch, C.M. and Elder, C.H., 1972. The direct method of determining methane content of coalbeds for ventilation design, U.S.B.M., Rep. Inv. 7767, 17 pp.
- Kissell, F.N., 1972. Methane migration characteristics of the Pocahontas No. 3 coalbed, U.S.B.M., Rep. Inv., 7649, 19 pp.
- Kissell, F.N., 1973. The methane migration and storage characteristics of the Pittsburgh, Pocahontas No. 3 and Oklahoma Hartshorne coalbeds, U.S.B.M. Rep. Inv. 7768, 23 pp.
- Lama, R.D., 1980. Drainage of methane from the solid at West Cliff Colliery, C.S.I.R.O. Geomechanics of Coal Rep. No. 18.
- Lama, R.D., Griffiths, L. and Marshall, P., 1980. Methane drainage investigations as a method of control of outbursts at West Cliff Colliery, Aus. I.M.M. Symposium on Prediction and Control of Outbursts in Coal Mines, Brisbane, Sept.
- Lama, R.D., 1980. Personal communication, April.
- Moffat, D.H. and Weale, K.E., 1955. Sorption of coal by methane at high pressures, Fuel, v. 34, no. 4.
- Mordecai, M. and Morris, L.H., 1974. The effects of stress on the flow of gas through coal measure strata, The Min. Engr., July, p.435.
- Mucke, G. and Koppe, U., 1975. Methane desorption within the area of influence of workings, Internat. Mine Vent. Congr., Sept., 1975, p. 125.
- Pickering, A.J., 1969. Air flows and methane emissions on fully caved, mechanised longwall faces in South Nottinghamshire, The Min. Engr., v. 129, p. 95.
- Richards, M.J. and Morris, L.H., 1978. Investigations into firedamp emission with reference to the coal face, The Min. Engr. June, p. 695.
- Somerton, H., 1975. Effect of stress on permeability of coal, Internat. J. Rock Mech. and Min. Sci., v. 12, p. 129.
- Stewart, I.M. 1971. Diffusional analysis of seam gas emissions in coal mines, Canad. I.M.M. Bull., April, p. 62.
- Thakur, P.C. and Davis, J.C., 1977. How to plan for methane control in underground coal mines, Min. Engg., Oct., p. 41.
- Zabetakis, M.G., 1972. Methane emission in coal mines: effects of oil and gas wells, U.S.B.M. Rep. Inv., 7658, 10 pp.

AR-124

APPENDIX C

RESERVOIR ENGINEERING IN COAL SEAMS:

PART 1 - THE PHYSICAL PROCESS OF GAS STORAGE
AND MOVEMENT IN COAL SEAMS

PART 2 - OBSERVATIONS OF GAS MOVEMENT IN COAL SEAMS

GRAY, I.; 1987

Reservoir Engineering in Coal Seams: Part 1—The Physical Process of Gas Storage and Movement in Coal Seams

Ian Gray, SPE

Summary. This is the first of two papers concerning the movement of gas in coal seams. It deals directly with the physical behavior of the coal seam as a reservoir. Coal seams show considerable differences in behavior from normal porous gas reservoirs in both the mode of gas storage and permeability characteristics. Most of the storage of gas in coal is by sorption into the coal structure, while the coal permeability is cleat-(fracture-) or joint-controlled and may vary over a wide range during production. This permeability fluctuation is not solely a phase relative permeability effect, but is rather a result of the opposing effects of effective stress increase with fluid pressure reduction and shrinkage of the coal. Reducing fluid pressure tends to close the cleats, reducing permeability, while shrinkage tends to open them.

Introduction

The two papers in this series evolved out of 3½ years of work (1979–82) on the problems of outbursting in Australian underground coal mines. This phenomenon involves the expulsion of gas and coal from the working face with resultant danger to the mining crew. Because energy release studies showed the outbursts to be primarily gas-driven,¹ efforts concentrated on studying gas drainage to alleviate the problem. Because underground mining methods were being used, most of the experimental work was conducted horizontally in-seam rather than from surface-drilled holes.

The mines studied are situated in the Bowen basin area, which extends from central to north Queensland, Australia. The seams with high gas contents are Permian age (260 million years) and bituminous, with rank determined by vitrinite reflectance lying between 1.00 and 1.30. The working depths were typically 380 m [1,247 ft] for Leichhardt Colliery, Blackwater; 135 m [443 ft] for Moura No. 4 mine, Moura; and 250 to 350 m [820 to 1,148 ft] for Bowen No. 2 Colliery, Collinsville. The first two mines contained a seam gas consisting predominantly of methane, while the Bowen No. 2 seam gas was mostly CO₂ of igneous origin.² In all three mines, seam thickness was about 6 m [20 ft], with seam slope lying between horizontal and 10° [0.17 rad] above horizontal.

Observations of flow and pressure variations in the seam caused by drainage into boreholes or mine openings showed that conventional oil and gas analysis approaches to describe the reservoir were inappropriate, particularly with respect to the apparent large increases in permeability with drainage. Attempts at simulation with a conventional simulator incorporating phase relative permeability effects could not explain large flow increases

measured during drainage. Conversely, some Japanese coal mines³ showed the opposite effect with apparently "self-sealing" coals.

The theory presented in this paper helps to explain the variations of permeability in coal associated with drainage.

Gas Storage in Coal

Gas is stored primarily by sorption into the coal. This typically accounts for 98% of the gas within a coal seam, depending on the pressure at which the gas is sorbed. In addition, gas is stored in the pore or cleat space either free or in solution. As Fig. 1, a typical sorption isotherm, shows, the amount of gas sorbed per unit increase in pressure decreases with increasing sorption pressure. Because gas drainage is often conducted under vacuum, the sorption isotherm has been extended below atmospheric pressure. Such tests for sorption may be conducted volumetrically⁴ or indirectly by weighing pressurized coal samples at a measured equilibrium pressure.⁵

Because seams are often water-saturated, in some cases it is possible that the water pressure exceeds the pressure at which all gas becomes sorbed into coal solids or into solution gas. This is the coal-seam equivalent of the bubble-point of an oil/gas system and is referred to in this paper as the equivalent sorption pressure.

Darcy Flow and Diffusional Movement

A number of references indicate that the movement of gas in coal is caused by Darcy flow down a pressure gradient or diffusion along a concentration gradient.^{6,7} The literature provides no clear answers as to which type of behavior is taking place or which type of transport governs the rate of gas production. One concept is that diffusive flow from the solids between cleats and Darcy flow along the cleat structure take place. This view, however, is not

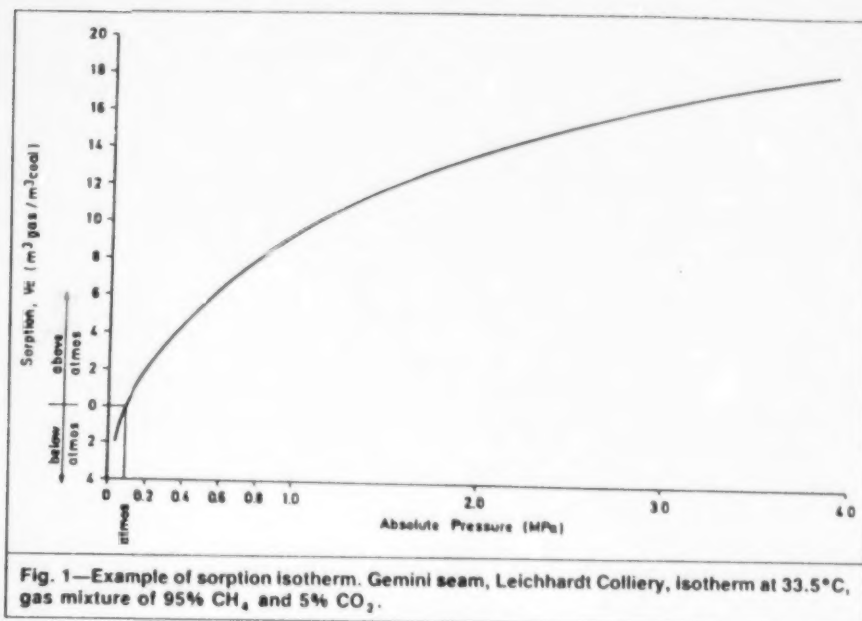


Fig. 1—Example of sorption isotherm. Gemini seam, Leichhardt Colliery, isotherm at 33.5°C, gas mixture of 95% CH₄ and 5% CO₂.

universal: Hemela* stated that tests on core showed diffusive behavior, while Tabor *et al.*⁸ measured core permeability changes with different saturations, suggesting Darcy behavior, and Somerton *et al.*⁹ measured Darcy behavior.

Observation of Australian coals indicates that the most important mode of fluid transport is Darcy flow in the cleats. The reasons for this are the close spacing of the cleats (typically 20 to 150 mm [0.8 to 5.9 in.]), the fact that most flow was measured as occurring along the cleats, the generally low bulk permeability of the coal (1.5 md), and the ready desorption of core in gas-content tests. The last involves taking coal core from borehole, placing it in a container, and measuring the gas evolved as described.¹⁰ Such core desorbed relatively quickly without a residual gas content. Such a residual gas content may be released from some U.S. coals only if the coal core is ground. The days or weeks taken for desorption of core drilled to approximately the same diameter as the cleat spacing is a short time interval compared with the months or years necessary for a coal block between boreholes to degas, which supports the hypothesis that Darcy flow is the rate-governing part of the drainage process for Australian coals. Other coal seams with a less fractured structure and with slower desorption from solid may show a rate that is controlled by diffusive movement. Because some of the blocky coals tend to be more porous in nature, however, the flow may shift back to being Darcian with gas movement through the pore structure, as in a more conventional reservoir.

As described in Part 2 of this paper, the permeability can be strongly directional-controlled by predominant cleat sets. This also lends weight to the concept that Darcian flow is of prime importance in the movement of gas within the coal.

Taking quantitative account of diffusive movement within a seam is difficult because the structure of the seam

could be considered to approximate a series of solid blocks between cleats. Because cleat spacing varies over a wide range—from a few millimeters to tens of meters—and the nature of the cleats varies significantly, it is virtually impossible to assign a mean cleat spacing and permeability into which the solid blocks could be considered to diffuse.

Variations in Coal Permeability

The permeability of the cleat structure within the coal may vary in two basic ways. The first is by phase relative permeability effects, whereby the degree of saturation will affect the gas and water relative permeabilities of the reservoir. This effect is well-known in the oil and gas industry. The second and probably more important way by which the permeability may vary is by a change in the effective stress within the seams. The effective stress is the total stress minus the seam fluid pressure. The effective stress tends to close the cleats and to reduce permeability within the coal. Although not proven conclusively by field and laboratory tests, it is very likely that the permeability is related particularly to the effective normal stress across the cleats because these appear to conduct most seam fluids. The effective normal stress referred to is the total stress normal to the cleat minus the fluid pressure within the cleat. If this is the case, then permeability variations brought about by variations in fluid pressures will be anisotropic, depending on the nature, frequency, and direction of the cleats. Such opening and closing of the cleats is also likely to change the phase relative permeabilities and capillary pressures within the coal.

The importance of effective stress is emphasized in this paper because of its large range (caused by seam drainage) compared with total stress and the large variations in permeability that may be brought about by changes in effective stress.

Relation Between Effective Stress and Permeability.

The effect of such a variation in permeability of coal core with confining stress was presented by Somerton *et al.*⁹

*Personal communication, M. L. Hemela, Broken Hill Proprietary Co. Ltd., Oil and Gas Div., Melbourne, Australia, May 12, 1982.

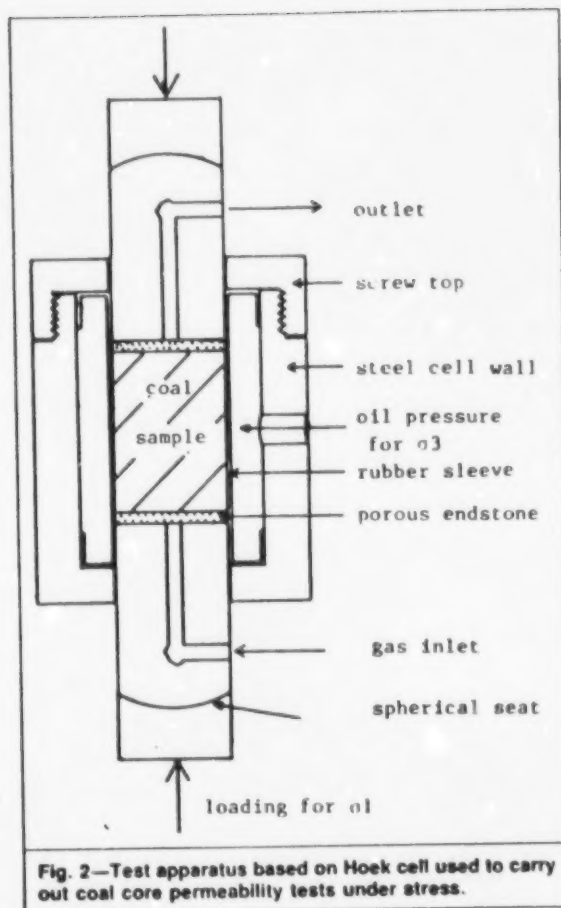


Fig. 2—Test apparatus based on Hoek cell used to carry out coal core permeability tests under stress.

Work at the Australian Coal Industry Research Laboratories confirmed their findings and showed that the relationship existing between effective stress and permeability for core taken from Leichhardt Colliery is approximately as shown in Eq. 1.

$$k = 1.013 \times 10^{-0.31\sigma} \quad (1)$$

where k is the dry permeability in md and σ is the effective triaxial confining stress in MPa. This suggests a 10-fold reduction in permeability with a 3-MPa [435-psi] increase in effective confining stress.

Note that the term "effective triaxial confining stress" is used here rather than "effective normal stress across cleats." The reason is that the core was tested for permeability in a triaxial Hoek cell,¹¹ and specific cleats generally could not be isolated within the 50-mm [2-in.] diameter cores. In addition, it was impossible to identify how the gas traveled through the specimen. Fig. 2 shows the test apparatus, and Fig. 3 shows test results from which this relation was established. The test procedure involved confining a core sample under a number of isotropic stress levels in the cell and passing methane

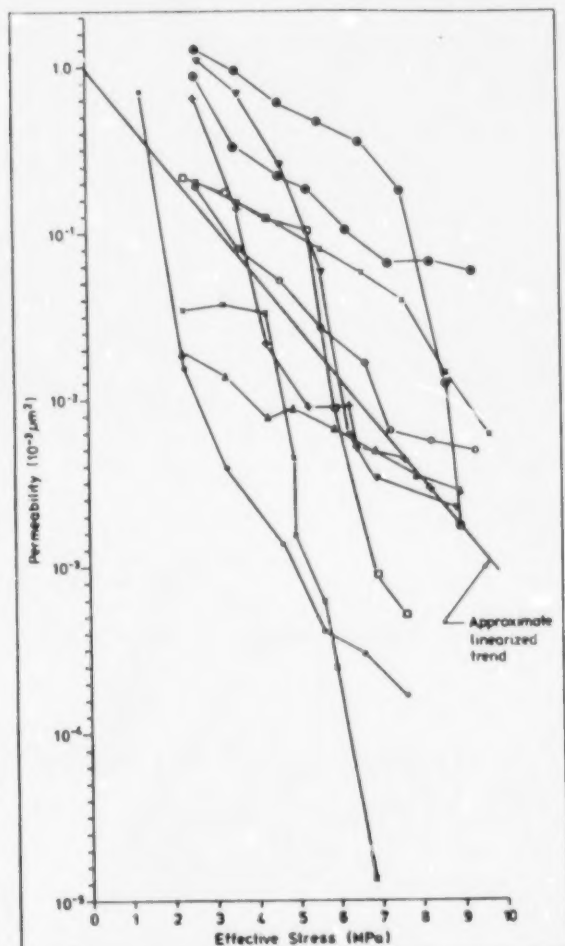


Fig. 3—Permeability vs. effective stress using CH_4 for core samples taken from the Gemini seam, Leichhardt Colliery.

through the sample at a steady pressure until the flow came to equilibrium. Because gas pressure varied across the specimen, the average of the two end gas pressures was used in the calculation of the effective stress. Eq. 2 was used to calculate core permeability.

$$k = (1.013 \times 10^{13}) \frac{2qp_o L \mu}{A(p_i^2 - p_o^2)} \quad (2)$$

As can be seen in Fig. 3, different core samples gave widely differing permeabilities. This is normal for coal samples taken from different plies in the seam and emphasizes the need for field testing that includes the whole seam.

Three factors influence the effective stress: initial stress, fluid pressure changes, and the shrinkage/expansion characteristic of the coal matrix, which is related to the equivalent sorption pressure in the seam.

Initial Stress in Seam. The initial stress in the seam is difficult to estimate because it is directional and not easily measured from surface or underground because of the

weak and jointed nature of the coal. Total overburden stress is not of primary interest, because it will remain unaltered by drainage, provided that this is carried out over a wide area. Rather, the horizontal stresses are more important because they act across the subvertical cleats that conduct gas. Some idea of the stress pattern possibly may be gained from stress measurement in the rock surrounding the seam. This may be carried out either by overcoring from underground workings¹² or by hydrofracture.¹³ This stress may be approximately separated into two components. The first is caused by gravitational effects—i.e., an overburden stress and the associated lateral stress developed under conditions of no lateral strain and given in Eq. 3.¹⁴

$$\sigma_h = \frac{\nu}{1-\nu} \sigma_v \quad \dots \dots \dots (3)$$

The second component is the tectonically induced horizontal stresses that, apart from redistribution through creep, should theoretically be proportional to the moduli of the rocks, provided that horizontal strains are approximately equal, as could be expected in extensive sedimentary deposits.

Fluid Pressures. Initial fluid pressures may be measured by conducting packer tests or by installing grouted sensors in seam as described in Part 2 of this paper.

Fluid pressure changes in the seam typically are brought about in the short term by drainage works or by the creation of mine openings. In geological time, the fluid pressures are probably controlled by the overburden thickness and the hydrostatic pressure associated with fluctuating groundwater levels.

As already mentioned, the water pressure in the seam is not necessarily related to the equivalent sorption pressure. In some cases, significant lowering of water pressure may have to take place before the equivalent sorption pressure is reached and sorbed gas is released. Such fluid pressure reductions will be associated with an increase in effective stress.

Equivalent Sorption Pressure. Equivalent sorption pressure needs to be measured because it is related to the state of coal matrix expansion and contraction. The most usual method would be to obtain core and desorb it fully so that an estimate of gas content can be obtained that may then be related to effective sorption pressure through the sorption isotherm for the coal. A more direct but as yet untried method would be to isolate the seam with packers and to lower the water pressure within the drillstring until gas is released. This method would require the solution pressure of the gas in the borehole fluid to be equal to the equivalent sorption pressure in the coal. This may not be the case if there has been inadequate time for equilibrium to become established.

Where the fluid pressure is lowered below the equivalent sorption pressure, gas is desorbed from the coal. This typically results in a shrinkage of the coal matrix,^{3,15} leading to a reduction of effective horizontal stress. This could lead to a complete loss of horizontal stress and the further opening of the cleats caused by continued shrinkage of the matrix.

Coal Shrinkage Related to Equivalent Sorption Pressures. Coal has been shown to shrink on desorption of gas and to expand again on resorption. Hargraves¹⁵ conducted a series of tests on coal from Metropolitan Colliery, New South Wales, Australia, that showed an average linear strain of 0.00182 per MPa change in equivalent sorption pressure using CO₂ as the gas. Japanese workers showed a similar but lesser effect using methane on coal from the northern Ishikari coal field in Hokkaido.³ Their average value was 0.000125 linear strain per MPa, more than an order of magnitude less. The latter low value is significant, particularly in the context of the "self-sealing" coals encountered in this part of Japan.

To incorporate this shrinkage effect into the estimation of effective stress within the coal, it is necessary to examine the elastic relation between stress and strain changes within the coal. If $\Delta\sigma_z$ is the change in vertical stress and $\Delta\sigma_x$ and $\Delta\sigma_y$ are changes in the principal horizontal stresses, then it is possible to write Eq. 4 describing the change in strain, $\Delta\epsilon_x$, in the horizontal x direction by use of elastic theory.

$$\Delta\epsilon_x = \frac{1}{E} (\Delta\sigma_x - \nu\Delta\sigma_y - \nu\Delta\sigma_z), \quad \dots \dots \dots (4)$$

where E is Young's modulus of elasticity.

If the changes in horizontal stress are the same and there is no change in vertical stress, then it is possible to write Eq. 5, which relates horizontal stress and strain changes.

$$\Delta\sigma_x = \frac{E}{1-\nu} \Delta\epsilon_x \quad \dots \dots \dots (5)$$

In the case of coal shrinkage, there is no real overall horizontal strain of the coal, provided that the effective stress does not drop below zero, because the coal is confined laterally. Where effective stress does drop below zero, the coal may be expected to shrink away from the cleats, and localized lateral strain of the coal will occur. Until this happens, the stress change in the coal can be described by replacing the strain-change term in Eq. 5 with the free strain-change value that could be expected in coal subject to desorption, Δp_{ve} (p_{ve} refers to equivalent sorption pressure). This in turn may be related directly to the equivalent sorption pressure change, Δp_{ve} , and Eq. 6 can be written to describe the variation in horizontal stress with a change in equivalent sorption pressure.

$$\Delta\sigma_x = \frac{E}{1-\nu} \frac{\Delta\epsilon_{ve}}{\Delta p_{ve}} \Delta p_{ve} \quad \dots \dots \dots (6)$$

The effects of fluid pressure increasing effective stress and matrix shrinkage reducing it cause opposing changes in permeability. These changes can be of several orders of magnitude. Where shrinkage is small, permeability may drastically reduce during drainage, while in the case of a high shrinkage, the permeability will increase.

Estimation of Permeability When Effective Stress Is Negative. To examine the range of permeability when effective stress extends into the negative range, it has been

TABLE 1—DERIVATION OF PERMEABILITY CHANGES WITH DRAINAGE. EXAMPLE FROM TEXT.

Absolute Fluid Pressure (MPa)	Equivalent Sorption Pressure (MPa)	Effective Stress (MPa)	Effective Cleat Width (μm)	Permeability (md)
Case b: $\Delta\epsilon_{wc}/\Delta p_{wc} = -0.00091 \text{ MPa}^{-1}$				
4.2	3.8	2.6	—	0.158
3.8	3.8	3.0	—	0.118
3.5	3.5	2.216	—	0.208
3.0	3.0	0.908	—	0.529
2.653	2.653	0.0	6.21	1.013
2.0	2.0	-2.0	18.1	25.03
1.0	1.0	-1.0	36.3	201.64
0.5	0.5	-0.5	45.4	394.15
Case d: $\Delta\epsilon_{wc}/\Delta p_{wc} = -0.000125 \text{ MPa}^{-1}$				
4.2	3.8	2.6	—	0.158
3.8	3.8	3.0	—	0.118
3.5	3.5	3.151	—	0.106
3.0	3.0	3.403	—	0.089
2.0	2.0	3.907	—	0.063
1.0	1.0	4.411	—	0.044
0.5	0.5	4.662	—	0.036

Nomenclature

- A = area of core cross section, cm^2 [in.²]
 b = crack width, mm [in.]
 k = permeability, md
 L = length of core, m [ft]
 p_i = inlet pressure, MPa [psi]
 p_o = outlet pressure, MPa [psi]
 p_{wc} = equivalent sorption pressure, MPa [psi]
 Δp = pressure variation, MPa [psi]
 Δp_{wc} = change in equivalent sorption pressure, MPa [psi]
 q = flow at outlet pressure, m^3/s [ft^3/sec]
 s = spacing between cracks, mm [in.]
 V_c = sorption, $\text{m}^3 \text{ gas}/\text{m}^3 \text{ coal}$ [$\text{ft}^3 \text{ gas}/\text{ft}^3 \text{ coal}$]
 $\Delta\epsilon$ = change in strain, m/m [ft/ft]
 $\Delta\epsilon_{wc}/\Delta p_{wc}$ = strain in unconfined coal specimen caused by a change in equivalent sorption pressure, MPa^{-1} [psi^{-1}]
 $\Delta\epsilon_x$ = change in strain in the horizontal x direction, m/m [ft/ft]
 μ = viscosity, Pa·s [cp]
 ν = Poisson's ratio
 σ = effective triaxial confining stress, MPa [psi]
 σ_h = horizontal stress caused by self weight, MPa [psi]
 σ_v = vertical stress caused by self weight, MPa [psi]
 $\Delta\sigma_x, \Delta\sigma_y$ = changes in the principal horizontal stresses, MPa [psi]

Acknowledgments

Most of the underground and laboratory work supporting the conclusions was conducted at the Australian Coal Industry Research Laboratories Ltd. During this period, the work was funded by the Australian Natl. Energy Research Development and Demonstration Committee, the

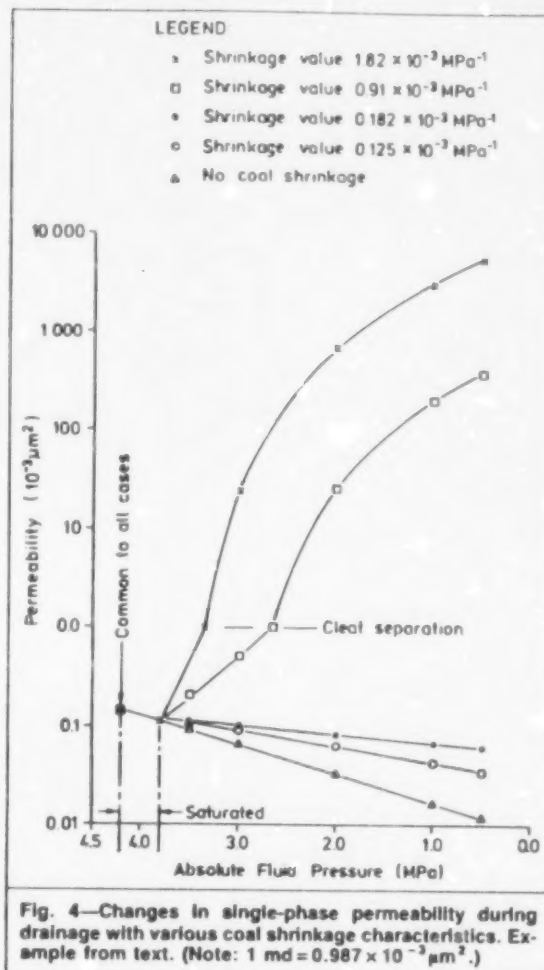


Fig. 4—Changes in single-phase permeability during drainage with various coal shrinkage characteristics. Example from text. (Note: 1 md = $0.987 \times 10^{-3} \mu\text{m}^2$.)

Australian Coal Assn., and the mining companies in whose mines the field work was conducted, including Collinsville Coal Co. Pty. Ltd., BHP Minerals Ltd., and Capricorn Coal Management Pty. Ltd. I thank these organizations for their support of this work.

References

- Gray, I.: "The Mechanism of and Energy Release Associated with Outbursts," *Proc., Symposium on the Occurrence, Prediction, and Control of Outbursts in Coal Mines*, Australasian Inst. of Mining and Metallurgy, Southern Queensland Branch, Brisbane (Sept. 1980) 111-25.
- Gould, K.W. and Smith, J.W.: "Isotopic Studies of Geochemical Factors in Outbursting," *Proc., Symposium on the Occurrence, Prediction and Control of Outbursts in Coal Mines*, Australasian Inst. of Mining and Metallurgy, Southern Queensland Branch, Brisbane (Sept. 1980) 85-98.
- Gray, I.: "Overseas Study of Japanese Methane Gas Drainage Practice and Visits to Coal Research Centres, June-August 1980," Australian Coal Industry Research Laboratories Ltd., Sydney, Report No. P.R. 80-15 (1980).
- Gray, I. and Truong, D.: "Methods and Equipment Used in Outburst and Gas Drainage Investigations," Australian Coal Industry Research Laboratories Ltd., Sydney, Report No. P.R. 82-9 (1982).
- Lama, R.D. and Bartosiewicz, H.: "Determination of Gas Content of Coal Seams," *Proc., Symposium on Seam Gas Drainage with Particular Reference to the Working Seam*, Australasian Inst. of Mining and Metallurgy, Southern Illawarra Branch, Wollongong (May 1982) 36-49.

assumed that all gas flow takes place in the cleats. The permeability can then be estimated by laminar flow theory for a series of parallel cracks, perpendicular to the direction of flow as described by Snow¹⁶ and shown in Eq. 7.

$$k = (1.013 \times 10^9) \frac{b^3}{12s}, \dots \dots \dots (7)$$

where b is the crack width and s is the spacing between cracks.

Thus if we know the permeability as zero effective stress is reached in the cleat, it is possible to estimate the cleat width for a given cleat density ($1/s$). Variations in this width may be calculated with further desorption and fluid pressure reduction. In each case, the shrinkage caused by desorption or the elastic expansion caused by a fluid pressure drop across the intercleat width (s) is added or subtracted, respectively, from the cleat width. For the example below, the contraction of the solid coal between cleats is assumed to take place without constraint perpendicular to the cleat set. This is reasonable because usually a secondary cleat set will intersect the principal set. The elastic strain, $\Delta\epsilon$, of each intercleat block associated with an isotropic change in effective stress caused by a fluid pressure variation Δp can be found from Eq. 8 (which is derived from Eq. 4). It is not known whether this type of elastic strain occurs with a block subject to different gas pressures because the nature of the effective stress in a sorbent medium as opposed to a porous medium is uncertain. In the case where water is the seam fluid, however, this type of relation seems reasonable.

$$\Delta\epsilon = \frac{1}{E}(1-2\nu)\Delta p. \dots \dots \dots (8)$$

Fortunately, the strain derived from testing samples of coal that are free to move in all directions will be the result of the shrinkage and effective stress terms and can be directly used in calculations where strain is related to varying equivalent sorption pressure.

Note that, while a zero or negative effective horizontal stress may exist in one direction, a positive value may exist in another because of the different initial in-seam stresses. The vertical effective stress will always be positive.

Example of Varying Seam Permeability With Drainage. To illustrate the effects of stress and shrinkage changes on the permeability of the coal to a single fluid, an example is given with assumed but reasonable values for seam properties. Incorporated into this example is a series of different possible shrinkage cases; the greatest was that recorded by Hargraves¹⁵ for Metropolitan Colliery coal, and the smallest was no shrinkage with varying equivalent sorption pressure. The relationship between effective stress and permeability used is that given in Eq. 1. Note that because of shrinkage the actual stress in seam will become anisotropic. This was not the case for the laboratory testing cited in the derivation of Eq. 1.

Seam parameters used for the example are (1) isotropic initial total stress in seam of 6.8 MPa [986 psi], (2)

initial seam water pressure of 4.2 MPa [609 psi], (3) initial equivalent sorption pressure of 3.8 MPa [551 psi], (4) absolute seam permeability at zero effective stress of 1.013 md, (5) Young's modulus of elasticity of 2.7 GPa [392×10^3 psi], (6) Poisson's ratio of 0.32, (7) cleat spacing of 20 mm [0.79 in.], and (8) seam shrinkage coefficients $\Delta\epsilon_{vc}/\Delta p_{vc}$ shown below (in MPa^{-1}).

Case a: -0.00182 Metropolitan coal with CO_2 gas;

Case b: -0.00091;

Case c: -0.000182;

Case d: -0.0000125 northern Ishikari coal with CH_4 gas;

Case e: 0.0.

The calculations for Cases b and d are given in Table 1, and the results for all cases are plotted in Fig. 4. As can be seen from this figure, very substantial variations in permeability take place because of changes in effective stress and the opening of the cleat structure below zero effective stress. Notably, as fluid pressure decreases without desorption, all permeabilities reduce. Then, depending on the shrinkage associated with desorption, the permeability may increase or continue to decrease as fluid pressures continue to drop along with equivalent sorption pressures. Where the effective stress falls to zero and cleat separation can occur, the permeability increases sharply.

Discussion

The concept of permeability variations closely controlled by effective stress—related both to fluid pressure and to shrinkage characteristics of the reservoir material—is substantially different from normal reservoir engineering concepts. Along with these permeability variations described here for a single phase, the other reservoir parameters—such as pore volume, capillary pressure, and phase relative permeability characteristics—can be expected to change. This makes calculating coal seam reservoir performance an extremely nonlinear and difficult problem.

One suggestion to overcome the permeability testing problems associated with sample strain is to stress core and then hold it at a constant circumferential strain during permeability testing. This would partially reflect conditions in seam. This type of test has limitations, however, because the main gas flow is usually parallel to the seam, not vertical to it, and principal stresses are probably different and unable to be modeled with conventional core permeability test apparatus that can provide only a single circumferential stress.

Conclusions

The storage and permeability characteristics of coal seams require that an approach substantially different from that for conventional gas reservoirs be used to assess their performance. Of primary importance is the storage of gas within the coal. Associated with this gas storage within the coal structure is a change in coal volume. Shrinkage of the coal occurs on desorption, leading to a effective stress reduction. This opposes the effective stress increase that would normally be expected with a lowering of fluid pressure. Because permeability is a function of the effective stress, it may increase or decrease with stress changes associated with drainage. Failure to acknowledge and take into account this effect will preclude reliable estimates of coal seam reservoir performance. These estimates are complicated additionally by seam inhomogeneity.

6. Cervik, J.: "Bureau of Mines Methane Control Program," U.S. Bureau of Mines, Technical Progress Report No. 10 (1969).
7. Hemela, M.L., Chapman, G., and Reichman, J.: "Mathematical Model for Simulation of Seam Gas Drainage," *Proc., Symposium on Seam Gas Drainage With Particular Reference to the Working Seam*, Australasian Inst. of Mining and Metallurgy, Southern Illawarra Branch, Wollongong (May 1982) 53-59.
8. Tabor, J.J. *et al.*: "Development of Techniques and the Measurement of Relative Permeability and Capillary Relationships in Coal," U.S.B.M., Pittsburgh, OFR 22-74 (1974).
9. Somerton, W.H., Soylomezoglu, I.M., and Dudley, R.C.: "Effect of Stress on the Permeability of Coal," *Int. J. Rock Mechanics, Mineral Science and Geomechanics Abstracts* (1975) 12, 129-45.
10. McCulloch, C.M. and Diamond, W.P.: "Inexpensive Method Helps Predict Methane Content of Coalbeds," *Coal Age* (June 1976) 81, No. 6, 102-06.
11. Hock, E. and Franklin, J.A.: "A Simple Triaxial Cell for Field and Laboratory Testing of Rock," *Trans., Instn. of Mining and Metallurgy*, London (1968) Sec. A, 77, 13-23.
12. Leeman, E.R.: "The Determination of the Complete State of Stress in Rock in a Single Borehole—Laboratory and Underground Measurements," *Int. J. Rock Mechanics and Mining Sciences* (1968) 5, No. 1, 31-56.
13. Enever, J.R. and Wooltorton, B.A.: "Demonstration of Hydraulic Fracturing Stress Measurement Techniques," CSIRO Australia, Div. of Geomechanics, Sydnal. Lecture Series 59 (1984) 1-57.
14. Hock, E. and Brown, E.T.: *Underground Excavations in Rock*, Instn. of Mining and Metallurgy, London (1980) 95.
15. Hargraves, A.J.: "Instantaneous Outbursts of Coal and Gas," PhD dissertation, U. of Sydney, Sydney, Australia (1963).
16. Snow, D.T.: "Rock Fracture Spacings, Openings and Porosities," *J. Soil Mechanics and Foundation Engineering Div.*, 94, No. 1, 73-91.

SI Metric Conversion Factors

$$\begin{array}{lll} ^\circ\text{F} & (^{\circ}\text{F}-32)/1.8 & = ^\circ\text{C} \\ \text{psi} \times 6.894\,757 & \text{E}+00 & = \text{kPa} \\ \text{psi}^{-1} \times 1.450\,377 & \text{E}-01 & = \text{kPa}^{-1} \end{array}$$

SPERE

Original manuscript (SPE 12514) received in the Society of Petroleum Engineers office Aug. 1, 1983. Paper accepted for publication May 8, 1985. Revised manuscript received March 31, 1986.

Reservoir Engineering in Coal Seams: Part 2—Observations of Gas Movement in Coal Seams

Ian Gray, SPE

Summary. This paper, the second of two concerning the movement of gas in coal seams, covers observations of seam fluid pressures and flows in mines in northern and central Queensland, Australia. Techniques based primarily on underground measurement rather than measurements from surface boreholes were used to gain information on the seams. The techniques used for in-seam studies are described because they differ substantially from conventional oil and gas surface borehole techniques. The paper demonstrates the importance of cleats and joints in the control of fluid movement and records flow increases consistent with increasing permeability with production.

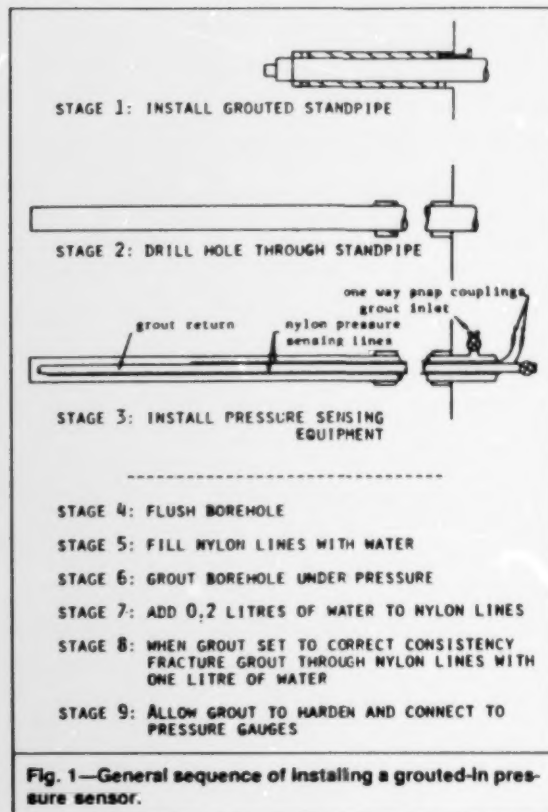
Methods

Pressure Measurement. Pressure measurement was conducted in seam by two basic techniques. In the first, a hole approximately 50 mm [2 in.] in diameter was drilled in seam and then the drill rods were withdrawn after a predetermined length, typically when 10 to 80 m [33 to 262 ft], had been reached. Following this, a 1- to 2-m [3.3- to 6.6-ft] -long packer was installed as quickly as possible on the end of the conduit string, leaving 2 to 10 m [6.6 to 33 ft] of open hole ahead of the packer. The packer was inflated with water through a synthetic braid hydraulic hose. Packer inflation pressure typically was raised to that required to expand the packer to borehole size plus $1\frac{1}{2}$ times the fluid pressure being measured. Occasionally, this packer pressure may have led to some problems because it may have approached or exceeded total in-seam stress and therefore tended to open the cleats (fractures) within the seam, thus promoting leakage around the packer. The pressure within the zone at the end of the packer was measured through a 4.8-mm [0.19-in.] -OD nylon tube leading to the borehole collar and connected to either a pressure gauge or a chart recorder. The coal permeability was normally sufficiently low that no significant leakage occurred around the packer into the borehole, and reasonably reliable pressure measurements could be obtained.

By repeating the procedure outlined above, it was possible to gain an understanding of the pressure distribution that existed within the coal along the line of a borehole. Up to five tests at 10-m [33-ft] intervals could be conducted in a 7-hour shift with this technique. Attempts were made to extend the use of packers to permanent pressure measurement. A single packer was considered inadequate to measure the overall pressure distribution along a borehole, and therefore multiple packer assemblies were constructed containing pressure measurement ports between the packers as described by Lama *et al.*¹ These were expensive and difficult to maintain.

In addition, they were invariably lost through borehole collapse if installed for longer than 2 weeks. On one occasion, accidental deflation of the packers led to a dangerous ejection of the unit from the borehole. For these reasons, the packer system was replaced by a system of grouted-in pressure sensor points that were developed specifically for long-term in-seam pressure measurement. These sensors have been installed successfully against pressures of 3.82 MPa [554 psi] and gas flows of up to 1900 m³/d [67,098 ft³/D]. Some of them have been monitored continually for 3 years.

To install such a grouted-in sensor, the procedure shown in Fig. 1 is followed. This initially requires drilling a hole for a standpipe (wellhead) approximately 130 mm [5.1 in.] in diameter to approximately 10 m [33 ft] in length. Within this, a 50-mm [2-in.] pipe is grouted and the hole is drilled on to 60 to 80 m [197 to 262 ft] at 50-mm [2-in.] diameter after the grout has set. Then 20-mm [0.8-in.] PVC conduit is installed up the borehole with 4.8-mm [0.2-in.] -OD nylon pressure-sensing lines strapped to it with their inner ends at pressure-sensing locations. Each end of nylon tube is wrapped with cloth to act as a filter. Once the sensor is fully installed, a faucet T, end cap, and valves are fitted to the standpipe to permit grouting. Before this, the borehole is flushed with water to remove loose coal particles. Natural gas flow can then be used to flush water from the hole. Just before grouting, the nylon lines are filled with water through one-way quick connections to prevent the ingress of grout within the lines. Grouting is accomplished up the annulus and back down the conduit return using a cement grout of 0.5 water/cement ratio containing calcium chloride and a post-initial-set expansion agent to promote a quick set and to ensure a tight seal. The grouting is completed at sufficient pressure to compress any gas bubbles to a small size. After grouting, a small amount of water is added to the nylon sensing lines to ensure no grout ingress. After initial grout set, an additional liter of water is injected into each line to fracture the grout and create an opening to the seam.



Once the grout has set, these lines can be connected to pressure gauges or to a chart recorder.

Flow Measurement. Flow measurement involves both gas and water and ideally should be conducted so that the flow contribution from each part of a borehole's length is measured. This can be done, but not on a continual basis because of the large effort required. It is more usual, therefore, to measure flows from the entire borehole

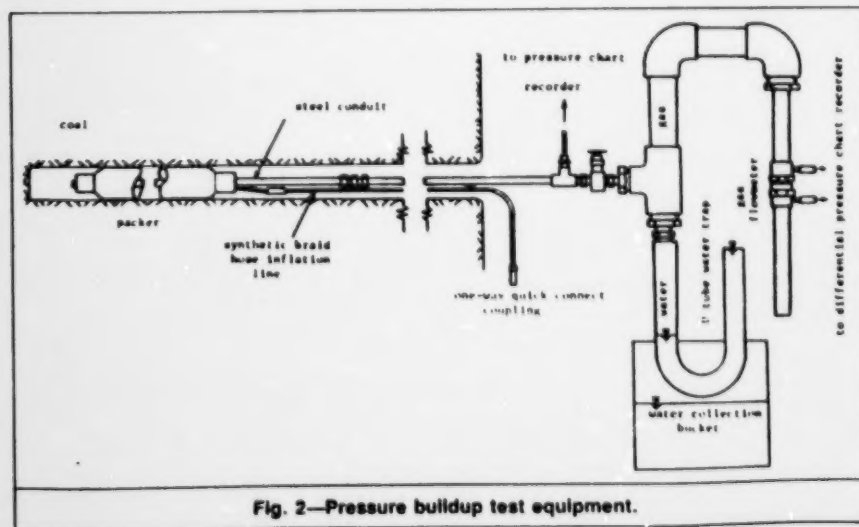
length on a daily basis and to carry out incremental flow tests to measure flow per unit length less frequently—e.g., monthly. It is possible to estimate the flow from each section of borehole by distributing the total borehole flow on the basis of the distributions measured in the incremental flow tests.

To measure total flow, a standpipe is grouted into the collar of each borehole; flow passes through this into a vertical pipe of sufficient diameter that the flow velocity is low and the water drains down into a collector tank, while the gas passes upward and through an orifice plate flowmeter.

For incremental flow testing, an inflatable packer is installed at the end of a conduit string to the base of the borehole and expanded. Flow is then ducted through the string and can be measured by an orifice plate flowmeter and water separator. If the packer is withdrawn incrementally and flow tests are conducted stage by stage, it is possible to derive a cumulative flow curve, the slope of which is the flow per unit length in the hole.

Combined Pressure and Flow Testing. Fig. 2 shows the equipment used to conduct buildup tests similar to those used in conventional reservoir engineering, where flow is permitted and then shut off, allowing pressure to build up. The buildup test has a slightly different form, because the borehole is flowing while it is drilled. To overcome this, the hole is shut in first and pressure allowed to build up. A flow period at atmospheric pressure follows with an additional buildup period. Typically, each flow or buildup period would last 1 to 2 days.

Gas Content Measurement. The usual method for measuring gas content in a coal seam is by coring the seam, retrieving the core as fast as possible, and permitting it to desorb in a canister. The gas evolved may then be collected and the desorption rate measured. This general procedure is described by McCulloch and Diamond.² Two aspects of this type of gas content determination should be described further. The first is the necessity of an estimate of the initial gas content lost before the core is in the canister. Normally, a plot of total evolved gas



vs. the square root of time since the start of gas evolution is made. This is supposed to be a straight line, and it is extrapolated back to zero time to find the total gas evolved. The definition of zero time requires some care, though, because the core will desorb gas at various rates, depending on water pressure within the borehole. In an in-seam hole, the core is expected to begin desorption immediately on drilling, while in a hole drilled from surface, the desorption will occur at various rates as the core rises in the borehole. The second aspect is that some workers have found it necessary to grind the coal to promote final gas loss within a reasonable time. However, this does not appear to be necessary for all coals.

Observations

Pressure Distributions Around Mine Openings. During the studies conducted in the underground mines of Queensland, pressure distributions were measured from mine openings with packers and grouted-in sensors. An example is shown in Fig. 3, a plan view of a section of Leichhardt Colliery mined in 1975 with pressure measurements made in boreholes drilled in 1981. Virgin seam pressure is 3.8 MPa [551 psi]. This case illustrates the importance of anisotropy caused by the cleat system. As can be seen, a high pressure gradient exists perpendicular to the cleat where no drainage can occur downcleat into the mine roadways. Where drainage can take place along the cleat into the roadway, the pressure gradient is low. This is reinforced by the cumulative incremental flow measurements made in Borehole C (across-cleat) and Borehole D (along cleat) and shown in Fig. 4. Borehole C shows substantial inflow along its entire length, with a major inflow at 31 m [102 ft] in length, presumably associated with some joint. Borehole D shows negligible inflow except where it traverses a differential compaction fault of 0.2-m [0.7-ft] throw. Some flow occurs in virtually all sections of Borehole C.

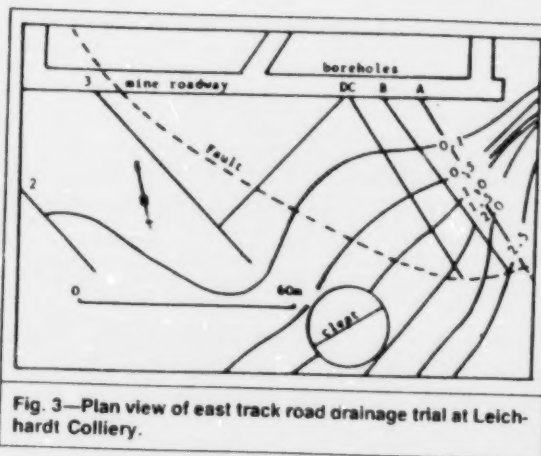


Fig. 3—Plan view of east track road drainage trial at Leichhardt Colliery.

Other examples of pressure distribution around openings typically tended to show a low-pressure-gradient zone initially, followed by a steep rise and flattening out of the pressure distribution to virgin seam pressure. The low-pressure distribution near the mine roadways often extends into solid coal substantially beyond the region where the stress redistribution caused by mining has any influence. This suggests an increasing permeability with drainage.

Drainage into Boreholes. To measure the effectiveness of a series of parallel boreholes drilled from a mine roadway into solid coal, a four-hole drainage pattern was used, as shown in Fig. 5. For a trial, two sets of drainage patterns were generally drilled, one across and one along the direction of the major cleat set. The outer two boreholes acted as cutoffs to flow laterally in seam. The inner two were expected to have a drainage behavior similar to that

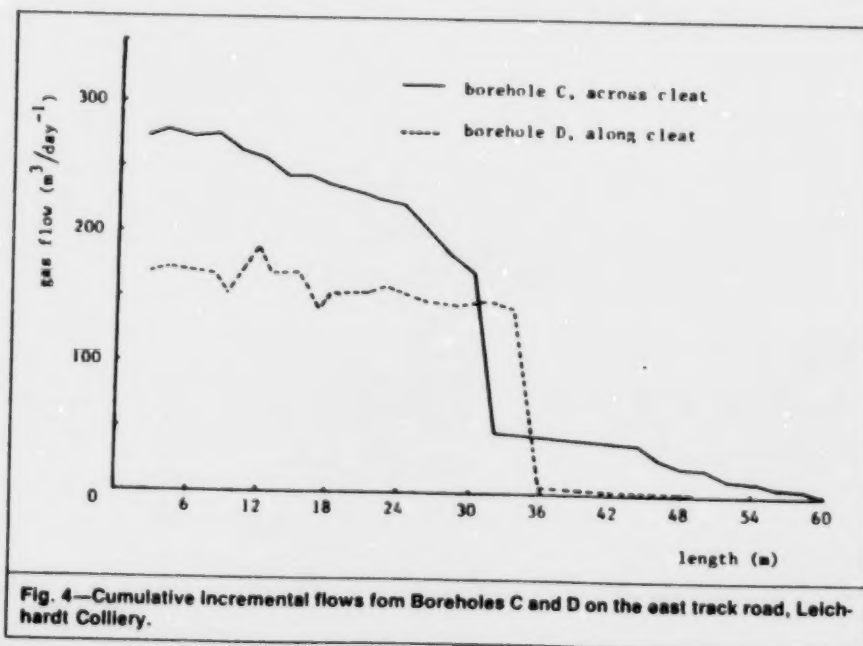


Fig. 4—Cumulative incremental flows from Boreholes C and D on the east track road, Leichhardt Colliery.

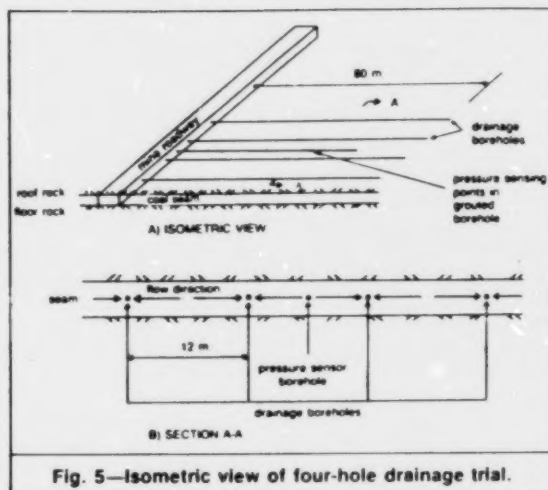


Fig. 5—Isometric view of four-hole drainage trial.

of a real drainage panel consisting of a series of boreholes drilled parallel to each other from a roadway. In the middle of the four drainage holes was a borehole fitted with a grouted-in pressure sensor. This system enabled the pressure to be monitored as drainage occurred. It was possible to carry out material-balance studies with this technique. Comparison of the gas content of the central block—derived by subtracting the quantity of gas drained from initial gas content measurements—allowed estimation of the quantity of gas left. This could be compared with estimates of gas content derived from pressure-sensor measurements and the sorption isotherm curves. Naturally, some drainage into the roadway would take place, as would drainage from virgin coal into the block. In the central area of the trial, however, a reasonable material-balance estimate usually could be made.

An example of such a material-balance estimate is shown in Fig. 6, which compares the two gas content estimates for an across-vein drainage panel in the Bowen No. 2 mine. As can be seen, a quite reasonable comparison of the two gas contents exists up to Day 220, when the gas content derived from pressure measurements and sorption isotherms tends to exceed that from a material balance. This indicates the presence of an external source, probably leakage from the surrounding stone and an adjacent seam.

The flow behavior of such holes is interesting because, in all the seams tested, the general pattern is that flow increases for a substantial period after the hole is drilled and then declines. Fig. 7 shows the measured average flow from the central region of inner boreholes in four-hole drainage patterns at the Bowen No. 2 and Moura No. 4 mines. In addition, the flow in Borehole B of the previously mentioned Leichhardt Colliery east track road is shown, though this is only the central borehole of a three-hole pattern into which substantial external leakage occurred. The general trend of increasing flow tends to support an increase in permeabilities with drainage.

Water flows occurred in all cases except those drilled in seams where substantial drainage had already taken place—such as Borehole D on the east track road at Leichhardt Colliery. The water flow could occur over the borehole length, as at Leichhardt Colliery, or be more associated with major joints, as was the case at Moura. In the Moura mine, the gas/water ratio of the central boreholes was initially less than 500 but rose to 6000 after 50 days' production. Outer boreholes tended to dry out less quickly.

Combined Pressure Buildup and Flow Tests. Fig. 8 shows the pressure buildups for the inner 10 m [33 ft] of a 56-mm [2.2-in.] borehole at Moura No. 4 mine. The

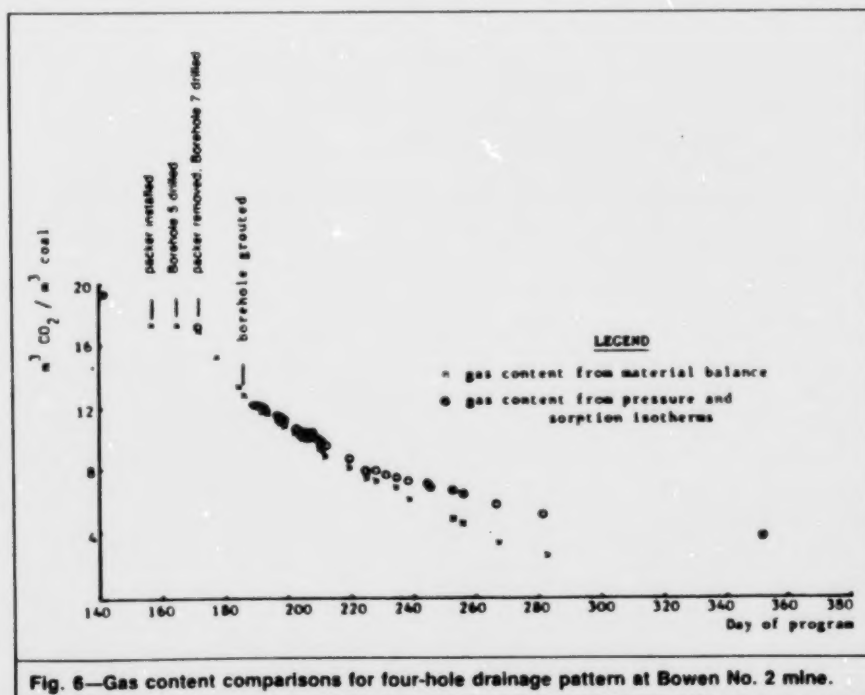


Fig. 6—Gas content comparisons for four-hole drainage pattern at Bowen No. 2 mine.

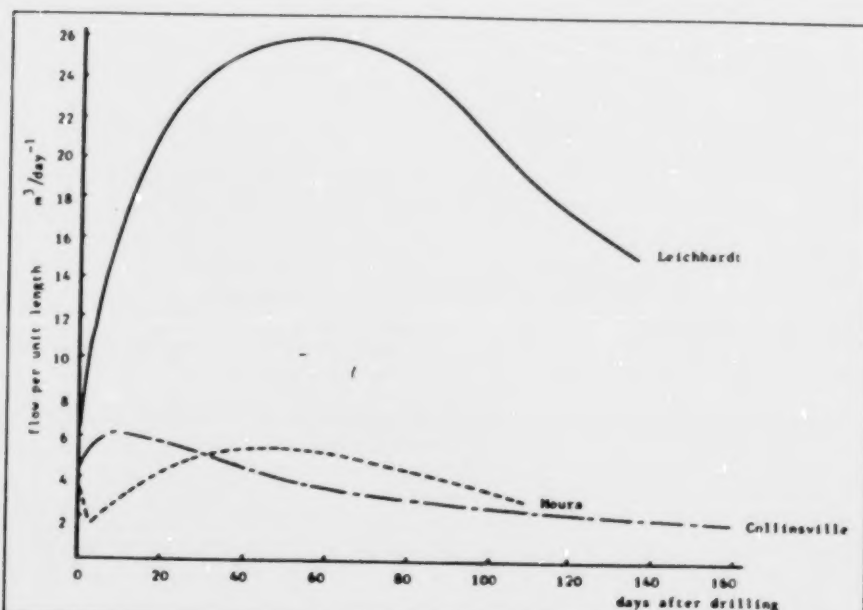


Fig. 7—Measured average flow per unit length for inner boreholes at Leichhardt Colliery, Moura No. 4 mine, and Bowen No. 2 mine.

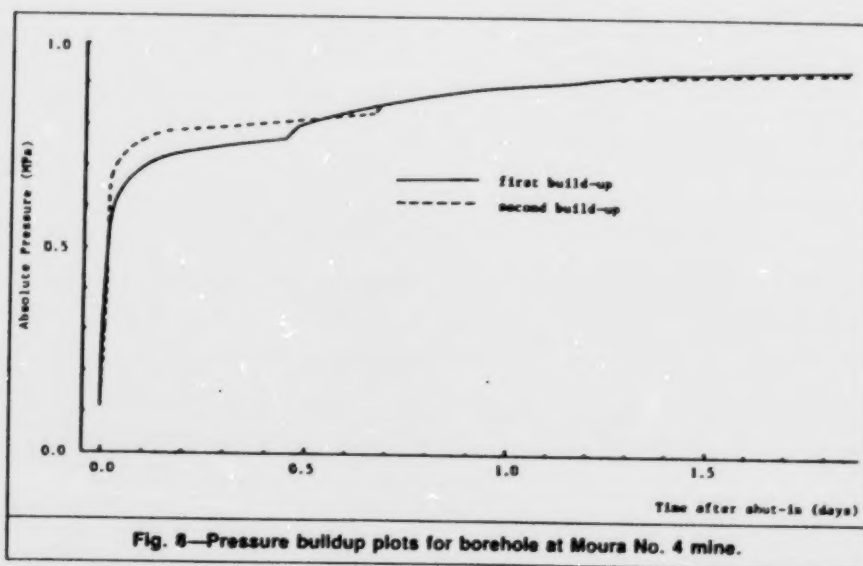


Fig. 8—Pressure buildup plots for borehole at Moura No. 4 mine.

first buildup period of 1.7 days follows 6.5 days of free flow after drilling. The second buildup follows an additional 1.06 days of drainage after the first buildup. As can be seen, the buildup is very steep initially and then flattens out sharply before an additional short rise after about ½ day, and then finally asymptotes to virgin seam pressure. The initial sharp rise is thought to be a result of intercleat coal blocks still desorbing into the cleats near the borehole, as opposed to the gas just being transported along the cleats. The sudden flattening out of the curve is probably associated with the gas pressure within the cleat exceeding the equivalent sorption pressure within the intercleat coal and therefore leading to gas being ab-

sorbed back into the coal. The small rise at ½ day after shut-in is thought to be associated with the filling of the borehole with water, thus stiffening the response characteristic of the system. The water flow before shut-in would have led to the borehole's filling with water in about ¼ day, so this hypothesis seems reasonable.

Seam Pressures and Gas Contents. Seam pressures and gas contents often seem to be related to the hydrostatic pressure at the level of the coal seam. In the case of Leichhardt Colliery, the measured seam pressure was 3.82 MPa gauge [554 psig] at 390-m [1,280-ft] depth. At Moura, the seam pressure was 1.1 MPa gauge [160 psig] at 135-m

[443-ft] depth. At German Creek mine in Queensland, seam pressures measured from surface also corresponded to hydrostatic pressure. At Bowen No. 2 mine, pressures appeared to be much lower than hydrostatic pressure, but because most of the methane there has been replaced by igneous CO₂ (which is much more easily transported in seam water), this is not surprising. Harrow Creek mine, also in the Bowen basin, has high water flows through the seam and virtually no gas remains; presumably the gas has been carried away in solution.

The gas contents obtained from core desorptions seem to be closely related to the seam fluid pressure through the sorption isotherm. This allows the hypothesis that as gas is evolved within the coal, it normally is released from sorption as the equivalent sorption pressure builds up and exceeds the hydrostatic head. If high water flows exist, however, it is probable that the seam gas has been removed in solution.

Conclusions

It appears that the cleating of coals is very important in the control of the principal directions of permeability. In addition to the cleats, major joints and faults within the seam act as conduits for seam fluid movement. From observations of the fluid pressure distribution around mine openings and the increase of flow from boreholes that occurs for a long period after drilling, it appears that permeability tends to increase substantially with drainage. Reasonable evidence suggests that gas contents in seams not subject to high water flows may be directly related to hydrostatic pressure through the sorption isotherm. In this instance, the equivalent sorption pressure is hydrostatic pressure.

Acknowledgments

Most of the underground and laboratory work supporting the conclusions was conducted at the Australian Coal Industry Research Laboratories Ltd. During this period, the work was funded by the Australian Natl. Energy Research Development and Demonstration Committee, the Australian Coal Assn., and the mining companies in whose mines the field work was conducted, including Collinsville Coal Co. Pty. Ltd., BHP Minerals Ltd., and Capricorn Coal Management Pty. Ltd. I thank these organizations for their support of this work.

References

1. Lania, R. D., Marshall, P., and Griffiths, L.: "Methane Drainage Investigations as a Method of Control of Outbursts at West Cliff Colliery," *Proc., Symposium on the Occurrence, Prediction and Control of Outbursts in Coal Mines*, Australasian Inst. of Mining and Metallurgy, Southern Queensland Branch, Brisbane (Sept. 1980) 223-39.
2. McCulloch, C. M. and Diamond, W. P.: "Inexpensive Method Helps Predict Methane Content of Coalbeds," *Coal Age* (June 1976) 81, No. 6, 102-06.

SI Metric Conversion Factors

ft	× 3.048*	E-01	= m
ft ³	× 2.831 685	E-02	= m ³
psi	× 6.894 757	E+00	= kPa

*Conversion factor is exact.

SPE

Original manuscript (SPE 14479) received in the Society of Petroleum Engineers office Aug. 1, 1983. Paper accepted for publication May 8, 1985. Revised manuscript received March 31, 1986.

AR-124

APPENDIX D

EFFECTS OF STIMULATION TREATMENTS ON
COALBEDS AND SURROUNDING STRATA

DIAMOND, W.P. AND OYLER, D.C.; 1987

Report of Investigations 9083

Effects of Stimulation Treatments on Coalbeds and Surrounding Strata

Evidence From Underground Observations

By William P. Diamond and David C. Oyster



UNITED STATES DEPARTMENT OF THE INTERIOR
Donald Paul Hodel, Secretary

BUREAU OF MINES
Robert C. Horton, Director

Library of Congress Cataloging-in-Publication Data

Diamond, W. P. (William P.)

Effects of stimulation treatments on coalbeds and surrounding strata.

(Report of investigations. 9083)

Bibliography: p. 45-46.

Supt. of Docs no.: I 28 23 9083

(1987)

1 Coalbed methane drainage 2 Boring 1 Oyler, David C II Title III Series Report of investigations (United States Bureau of Mines). 9083

TN23.U43 [TN305] 622 s [622'.8] 86-607907

CONTENTS

	Page		Page
Abstract	1	Upper Freeport Coalbed, Pennsylvania	27
Introduction	2	Lucerne No. 6 Mine, borehole RP-1	27
Concepts and limitations of underground observations	2	Lucerne No. 6 Mine, borehole RP-2	31
Borehole history and results of underground observations	4	Lucerne No. 6 Mine, borehole RP-3	33
<i>(Each of the sections listed below on individual boreholes includes subsections on borehole completion, treatment implementation, and underground observations.)</i>		Lower Kittanning Coalbed, West Virginia	36
Blue Creek Coalbed, Alabama	4	Kitt No. 1 Mine, borehole KE-2	36
Oak Grove Mine, borehole TW-1	4	Kitt No. 1 Mine, borehole DGBH-5	37
Oak Grove Mine, borehole TW-2	6	Illinois No. 6 Coalbed, Illinois	39
Oak Grove Mine, borehole TW-3	8	Inland Steel Mine, borehole 1-NE	39
Oak Grove Mine, borehole TW-4	10	Jawbone Coalbed, Virginia	40
Oak Grove Mine, borehole TW-5	12	McClure No. 1 Mine, borehole DG-1A	40
Oak Grove Mine, borehole DHM-5	15	Rock Canyon Coalbed, Utah	41
Oak Grove Mine, borehole DHM-6	16	Soldier Canyon Mine, borehole SC-1	41
Pittsburgh Coalbed, Pennsylvania	18	Summary of observations	42
Vesta No. 5 Mine, borehole USBM-4	18	Conclusions	45
Emerald Mine, borehole EM-5	20	References	45
Emerald Mine, borehole EM-6	21	Appendix.—Acquisition and limitations of stimulation treatment data	47
Emerald Mine, borehole EM-7	22		
Emerald Mine, borehole EM-8	22		
Emerald Mine, borehole EM-11	26		
Cumberland Mine, borehole CNG-1034	26		

ILLUSTRATIONS

1. Relationship of mine entry orientation and interception of vertical fracture wings parallel to coal cleat	3
2. Treatment record, borehole TW-1	4
3. Mine map, borehole TW-1, with fracture orientations	5
4. Schematic cross section parallel to face cleat fractures at borehole TW-1	5
5. Treatment record, borehole TW-2	6
6. Mine map, borehole TW-2, with fracture orientations	7
7. Cross section of coal face at borehole TW-2, with fracture orientations	7
8. Comparison of gamma density logs before and after jet slotting, borehole TW-2	8
9. Treatment record, borehole TW-3	9
10. Mine map, boreholes TW-3 and TW-4, with fracture orientations	10
11. Treatment record, borehole TW-4	11
12. Stratigraphic section and near-wellbore underground observations in the Mary Lee-Blue Creek Coalbed intervals, borehole TW-5	12
13. Treatment record, borehole TW-5	13
14. Mine map, borehole TW-5, with fracture orientations	14
15. Mine map, boreholes DHM-5 and DHM-6, with fracture orientations	16
16. Treatment record, borehole DHM-6	17
17. Treatment record, borehole USBM-4	18
18. Mine map, borehole USBM-4, with fracture orientations	19
19. Cross section of face cleat fracture in entry at location 1 near borehole USBM-4	19
20. Cross section of butt cleat fractures on rib at location 2 near borehole USBM-4	19
21. Map of Emerald Mine area with locations of boreholes EM-5, EM-6, EM-7, EM-8, and EM-11	20
22. Mine map, borehole EM-5, with fracture orientations	20
23. Cross section of fractures on rib closest to borehole EM-5	20
24. Treatment record, borehole EM-6	21
25. Mine map, borehole EM-6, with fracture orientations	22
26. Cross section of fractures on rib west of borehole EM-6	22
27. Treatment record, borehole EM-8	23
28. Mine map, borehole EM-8, with fracture orientations	24
29. Cross section of fractures on rib at location 1 near borehole EM-8	25
30. Cross section of fractures on rib at location 2 near borehole EM-8	25
31. Cross section of fractures on rib at location 3 near borehole EM-8	25
32. Cross section of fracture on rib at location 4 near borehole EM-8	25

ILLUSTRATIONS—Continued

	Page
33. Mine map, borehole EM-11, with fracture orientations	26
34. Cross section of fractures on rib near borehole EM-11	26
35. Mine map, borehole CNG-1034, with fracture orientations	27
36. Treatment record, borehole RP-1	28
37. Mine map, boreholes RP-1 through RP-3, with fracture orientations	29
38. Cross section of fractures at location 25 on the southeast rib near borehole RP-1	30
39. Mine map, boreholes RP-1 through RP-3, with horizontal distribution of paint on roof	31
40. Treatment record, borehole RP-2	32
41. Cross section of fractures at location 7 on the northwest rib near borehole RP-2	33
42. Mine map, boreholes RP-1 through RP-3, with horizontal distribution of paint and sand on top of shale parting	33
43. Treatment record, borehole RP-3	34
44. Cross section of fractures at location 15 on southeast rib and location 16 on northwest rib near borehole RP-3	35
45. Treatment record, borehole KE-2	36
46. Mine map, borehole KE-2, with fracture orientation	37
47. Treatment record, borehole DGBH-5	38
48. Mine map, borehole DGBH-5, with fracture orientations	38
49. Cross section of fractures on east and west ribs near borehole DGBH-5	39
50. Treatment record, borehole 1-NE	40
51. Mine map, borehole 1-NE, with fracture orientations	40
52. Treatment record, borehole SC-1	41
53. Mine map, borehole SC-1	42

TABLES

1. Specifications of gelled stimulation fluid, borehole TW-2	6
2. Summary of treatment chronology, borehole TW-5	13
3. Summary of fracture observations (paint-coated surfaces), borehole TW-5	14
4. Summary of treatment chronology, borehole DHM-6	17
5. Summary of treatment chronology, borehole USBM-4	18
6. Summary of treatment chronology, borehole CNG-1034	27
7. Summary of treatment chronology, boreholes RP-1 through RP-3	29
8. Orientation and description of vertical fractures, boreholes RP-1 through RP-3	30
9. Summary of treatment chronology, borehole 1-NE	40
10. Summary of stimulation treatment data	43
11. Summary of underground observations for 22 intercepted stimulation treatments	43

UNIT OF MEASURE ABBREVIATIONS USED IN THIS REPORT

bbl bbl min ft ft' ft' ³ gal h in lb lb/in ²	barrel barrel per minute foot cubic foot cubic foot per day gallon hour inch pound pound (force) per square inch	lb/in ² (abs) lb/in ² (ga) lb gal Mft ³ /d min MMft ³ pct rpm std ft ³ /min yr	pound (force) per square inch, absolute pound (force) per square inch, gauge pound (mass) per gallon thousand cubic feet per day minute million cubic feet percent revolution per minute standard cubic foot per minute year
---	---	--	---

EFFECTS OF STIMULATION TREATMENTS ON COALBEDS AND SURROUNDING STRATA

Evidence From Underground Observations

By William P. Diamond¹ and David C. Oyler²

ABSTRACT

Stimulated vertical boreholes are an effective means of removing gas from coalbeds in advance of mining. This Bureau of Mines report examines the coal mine roof damage potential of stimulation treatments. Twenty-two Government-sponsored stimulation treatments have been mined through to determine the effects on the coalbed and roof strata. Vertical fractures in the coalbed were discernible for most treatments and horizontal fractures were present for about half of the stimulations. Sand-propped vertical fractures were usually short in lateral extent. Evidence of stimulation fluid movement could generally be traced beyond the maximum extent of sand-filled fractures when fluorescent paint was added to the treatment fluids. The maximum lateral extent for a sand-filled vertical fracture was 416 ft and a paint-coated vertical fracture extended 630 ft. Horizontal fractures were generally found within bedding planes, most commonly on top of the coalbed. The maximum lateral extent for a horizontal fracture was 265 ft at a coalbed depth of 630 ft. Horizontal fractures have been found to occur as deep as 1,145 ft. Penetration of strata overlying coalbeds was observed in nearly half of the treatments intercepted. Most of these occurrences have been interpreted to be penetrations into preexisting joints. No roof falls or adverse mining conditions were encountered that could be attributed to the stimulations.

¹Geologist

²Mechanical engineer

Pittsburgh Research Center, Bureau of Mines, Pittsburgh, PA

INTRODUCTION

The Bureau of Mines has developed several techniques, including the use of horizontal and vertical boreholes, to remove gas from coalbeds in advance of mining. Horizontal boreholes drilled from underground workings as part of the mining cycle have been shown to be very effective in providing short-term, immediate relief from high methane emissions (1-6).¹ This technique has been widely accepted in the coal mining industry, but it does require close coordination to integrate the drilling and subsequent gas drainage and disposal into the projected mine development plan.

Vertical boreholes can be placed several years in advance of mining to predrain gas from coalbeds over relatively large areas (7-11). The vertical borehole technique has the additional advantage over horizontal boreholes of allowing work to be performed on the surface instead of in the more restrictive underground environment. However, except for the relatively large-scale vertical borehole programs for both mine safety and for commercial production in the Black Warrior Basin of

Alabama (7, 9, 11), the technique has been underutilized. The primary reason for this seems to be a combination of the economic conditions in the coal industry, legal questions as to the ownership of coalbed gas, and the potential of roof damage from the stimulation treatments. The question of potential roof damage is the subject of this report.

A total of 22 Government-sponsored stimulation treatments in coalbeds have been intercepted by mining. Twenty-one of these interceptions were in the eastern United States (10 in Pennsylvania, 7 in Alabama, 2 in West Virginia, and 1 each in Illinois and Virginia); only one in the western United States (Utah) was available for investigation. These underground interceptions have provided a unique opportunity to observe directly the actual effects of the stimulation treatments on coalbeds and surrounding strata. This report details observations previously presented in various forms by the Government and its contractors as well as information not heretofore reported, including the results of additional interceptions.

CONCEPTS AND LIMITATIONS OF UNDERGROUND OBSERVATIONS

Underground evaluation of areas surrounding boreholes that have been stimulated to increase gas production from coalbeds is an important step in obtaining information for the development of efficient stimulation treatments that do not adversely affect mining. By directly observing the effects of a particular stimulation design on a coalbed and surrounding strata and evaluating the observations in conjunction with thorough geologic characterization, valuable information can be obtained, especially early in the development stages of a methane drainage program. It is important to note that what is seen underground is dependent upon the area exposed by mining and the timing of observations in relation to the advance of entries. Obviously once a volume of coal is mined, anything contained in that coal is forever lost for direct examination.

Geologic characterization of available underground workings in the area to be drilled and stimulated is highly recommended before stimulation. This preliminary characterization aids in interpreting the underground observations and provides information for placement of the borehole and design of the treatment. These geologic studies should be continued as the entries are advanced toward the stimulated borehole and through the actual interception and advance beyond the borehole. The most complete characterization of this type was conducted in conjunction with Bureau of Mines contract J0333908, the final report (12) of which can serve as a model for future work.

The results of previously evaluated treatment mine-throughs indicate that it is quite likely that the predominant fracture wings will generally follow or penetrate the face cleat in the coalbed. The face cleat orientation can be determined by measuring cleat orientations underground if mine workings are available, or it might be inferred either from orientations in coalbeds

exposed at the surface or from analysis of surface rock joints (13-14). The identification, characterization, and orientation measurement of roof joints is also important since it appears that many roof penetrations have actually been invasions into preexisting roof joints.

The cleat and roof joint orientation data can be used in placing the location of boreholes in relation to the mine development plan. This in turn can affect what is seen underground and possibly the seriousness of any penetration of existing roof joints. A borehole to be evaluated underground could be drilled so that it would be located in an entry to be intercepted by mining, allowing direct observation of the borehole, which would potentially provide the maximum information. Alternatively it could be placed in a pillar where less information may be gained, but where any roof penetration (which might or might not result in roof instability) near the wellbore would potentially not extend to the developed entries around the pillar.

If a borehole is to be mined through, it may also be possible to place it so that the probable face cleat orientation for an induced fracture would be parallel or perpendicular to the long axis of the entry (fig. 1). A hydraulically induced fracture in the face cleat direction and oriented parallel (or nearly parallel) to the long axis of the entry would be encountered "head-on" by the advancing entry (borehole 1, figure 1). This would provide an opportunity to perhaps observe the entire length of the fracture from its tip back to the borehole, if it was possible to observe the advancing face continuously during mining. However, once mining of the entry was completed, the fracture would have been consumed and would no longer be available for evaluation.

Butt cleat induced fractures, which are commonly shorter and less extensive than those in the face cleat orientation, would perhaps be observable on the ribs adjacent to the borehole. Complications could occur if roof joints (or other linear geologic discontinuities) are present in the entry and if they are parallel to the orientation of

¹Italic numbers in parentheses refer to items in the list of references at the end of this report.

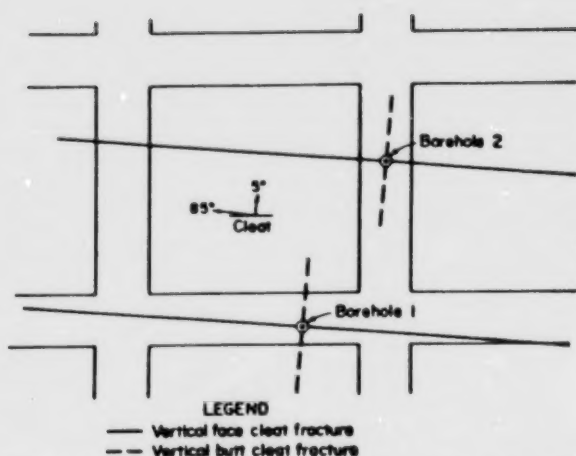


Figure 1.—Relationship of mine entry orientation and interception of vertical fracture wings parallel to coal cleat.

the face cleat, which commonly is the case. If the stimulation treatment penetrated into roof joints, resulting in roof instability, the hazard would potentially be exposed over a greater length in the entry if the joints paralleled the long axis of the entry than if the roof joint only extended across the width of the entry.

A hydraulic fracture trending in the face cleat direction perpendicular to the long axis of an entry would presumably be contained in the ribs adjacent to the borehole (borehole 2, figure 1) and would be available for observation as long as the pillar remained. Butt cleat fractures, if present, would have been intercepted head on. Since butt cleat fractures are often relatively short, they would often be completely mined by a single mining cut and never observed.

When the borehole is located in a pillar or other unmined area, there is always some uncertainty as to the exact location of the borehole. Knowledge of the exact location of the borehole is important in interpreting the underground observations, in terms of determining the length of the fracture wings and the presumed pathways (amount of stairstepping along face and butt cleats away from the borehole) of the treatment fluids through the coalbed. The uncertainty as to the borehole location results from the propensity for a borehole to deviate from vertical when drilled, and from inaccuracies in surveys of surface location, borehole deviation, and mine workings used to determine where the borehole should be underground.

Even with careful preplanning of the borehole location in relation to the expected underground location, it may not end up where it is expected. This was observed in the case of borehole SC-1 at the Soldier Canyon Mine in Utah. The bottom hole location of this borehole (as discussed in a later section of this report) was determined from a deviation survey to have migrated 35 ft laterally from the surface location. This should have placed it near the middle of an entry according to the mine plan. A thorough inspection of this area after mining failed to reveal any trace of the borehole or of the sand proppant and stimulation fluids, suggesting that one or more of the surveys must have been in error.

A significant aid in locating and describing the pathways of fracturing fluid movements underground is

the use of fluorescent paint in the treatment fluids. By using a permissible ultraviolet light underground, the fluorescent paints can usually be readily identified and the extent of their pathways mapped. In general, when the fluorescent paint has been used, it has been possible to trace the movement of the fracturing fluids significantly beyond the extent of sand-propped fractures. Many different paint colors have been used, with the best results observed from orange, yellow, green, and red paint.

It should be noted that coalbeds sometimes contain naturally occurring fluorescent materials that may confuse the underground observations if paint similar in color to the materials are used. Red and pink fluorescing minerals were observed in the Upper Freeport Coalbed at the Lucerne No. 6 Mine in Pennsylvania, where red paint was used along with several other colors. Green fluorescing resins were observed in the Rock Canyon Coalbed at the Soldier Canyon Mine in Utah where green was used as the only paint color in the treatment. Blue fluorescent paint can be a particular problem, as was seen at the Lucerne No. 6 Mine, where hydraulic fluids used in the mining machinery also fluoresced blue making it difficult to differentiate the two.

During the stimulations at the Lucerne No. 6 Mine, different paint colors were injected at various stages of the treatment in an attempt to differentiate the effects of the different stages underground. The results of this experiment were somewhat varied because of the tendency for the last paint injected along a particular fracture to cover and mask any paint injected earlier.

Care must be taken in comparing underground observations of fluorescing paint tagged treatments to treatments where paint was not added. This is especially important in comparing maximum fracture length. In 5 of the 22 mined through treatments, fluorescent paints were used along with sand and in each case where any fractures were seen (four of five), it was possible to map the fluid movements beyond the presence of sand proppant. Paint was quite often observed on or in both vertical and horizontal fractures. Although it was obvious that treatment fluids had flowed through these fractures, it was not possible to determine if they previously existed or if they had been created by the treatment. Except in a few cases most fractures follow paths that look like cleat and bedding planes, and the fractures may have existed before the treatment began.

Experience with wells where fluorescing paint has been added suggests that, in shallow stimulated coalbeds, the major sand-filled fractures in the face and butt cleat directions are paralleled by networks of planes along which fluids, but not sand, have traveled. In addition, the fluids may also stairstep short distances along face and butt cleats to form a general fracture trend that does not parallel the face or butt cleat direction. Multiple horizontal planes of fluid movement have been identified with the use of fluorescent paint. As many as four horizontal planes have been observed in one case, although it is more common to see only one or perhaps two planes.

Vertical fractures may also lie over and connect to horizontal planes. These occurrences seem to be of limited extent, indicating that they may be fractures actually created by the stimulation process and therefore require too much creation energy to cover large areas when preexisting vertical and horizontal paths for fluid movement already exist.

BOREHOLE HISTORY AND RESULTS OF UNDERGROUND OBSERVATIONS

The available borehole completion and stimulation treatment data for each of the 22 mined-through stimulation treatments are presented in the following sections along with the results of the underground observations. Data are grouped together for each coalbed treated to aid the reader in locating and interpreting the results. Summary tables of the most pertinent data are

included in the "Summary of Observations" section of this report. It should be noted that the coalbed mined and generally completed for gas production at the Oak Grove Mine is termed the Blue Creek Coalbed in this report, which is common usage in the mining district. Stratigraphically this coalbed is the bottom bench of the Mary Lee Coalbed (15-17).

BLUE CREEK COALBED, ALABAMA

OAK GROVE MINE, BOREHOLE TW-1 (18-19)

Borehole Completion

Borehole TW-1 was drilled 11 in. in diameter, using foam, to a depth of 1,115.25 ft and cased with 8 $\frac{1}{2}$ -in.-OD casing to a depth slightly less than 1,115 ft. The bottom of casing was placed approximately 2 to 2.5 ft into the top of the target Blue Creek Coalbed, which was later reported to be 6 ft thick. The casing was cemented in place with an expanding type 16-lb gal cement containing 18 pct NaCl. Approximate bottom hole pressure during cementing was 1,000 lbf/in²(ga) $\pm$ 100 lbf/in². After the casing was cemented in place, a 7 $\frac{1}{2}$ -in.-diameter tricone roller bit was used to drill out the float shoe on the bottom of casing and to extend the hole to a depth of 1,133.25 ft, approximately 15 ft below the base of the Blue Creek Coalbed. An 8- to 10-lb gal drilling mud was used for this short drilling interval, and the hole was circulated with clear water for several hours after the completion of drilling to clean the hole. At the time of completion, borehole TW-1 was approximately 520 ft ahead of mining.

Treatment Implementation (November 23, 1975)

The stimulation treatment record for borehole TW-1 is shown in figure 2. A discussion of the acquisition and

limitations of stimulation treatment data is given in the appendix. The treatment utilized gelled water and sand proppant. The stimulation designed was generated with computer-based techniques that used formation parameters to estimate the character of the Blue Creek Coalbed. A treatment was selected that would theoretically produce two vertical fracture wings, each extending 423 ft into the coalbed with a sand proppant wing length of 143 ft.

The borehole was stimulated through 2 $\frac{1}{4}$ -in.-diameter tubing set at 1,084 ft with a packer installed at the top of the wellhead to prevent the stimulation fluid from flowing to the surface in the annulus between the casing and tubing. No prestimulation cleanup or slotting of the coal face with a jetting tool was reported. The treatment was initiated with a 2,500-gal gelled water pad injected at rates averaging about 2 bbl/min. Surface treatment pressures quickly increased to about 1,400 lbf/in²(ga), leveled off (indicating leakoff into the formation), then gradually increased to 1,775 lbf/in²(ga) where formation break occurred with an accompanying sharp pressure decline to 600 lbf/in²(ga).

After the formation break, the injection rate was increased to 10.5 bbl/min. The surface pressure increased to nearly 1,000 lbf/in²(ga) and then gradually decreased to about 600 lbf/in²(ga) throughout the remainder of the pad injection. An additional 2,500 gal of gelled water

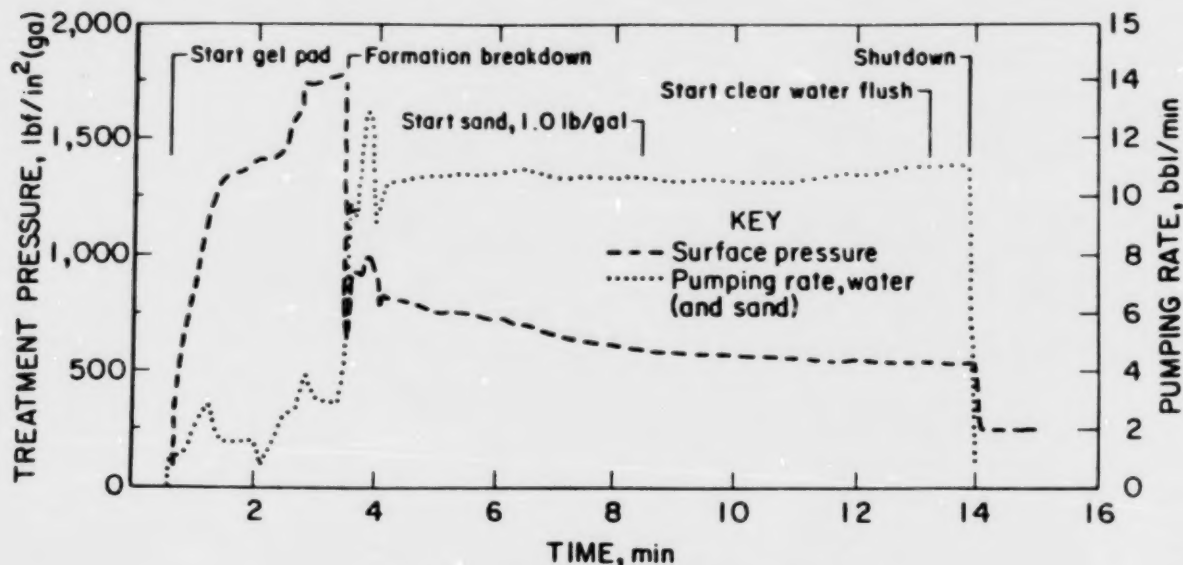


Figure 2.—Treatment record, borehole TW-1 (18).

containing 2,500 lb of minus 20- to plus 40- (20/40) mesh sand (at 1 lb/gal) was injected at a pump rate of 10.5 bbl/min. This was followed by a 290-gal fluid flush without sand. A total of 5,290 gal of fluid were injected during the treatment. Treatment pressures (surface) immediately after formation breakdown averaged about 650 lbf/in²(ga). Surface treatment pressures averaged about 550 lbf/in²(ga) during the sand injection stage of the treatment. An instantaneous shut-in pressure (ISIP) of 250 lbf/in²(ga) (surface) was recorded at the end of the treatment. The ISIP is a measure of the minimum pressure required to keep a fracture open. Gas production from borehole TW-1 began at about 2,500 ft³/d and gradually increased to a maximum of only 8,500 ft³/d just prior to underground interception.

Underground Observations

Borehole TW-1 was mined through on June 20, 1976, approximately 7 months after stimulation. The area of the borehole interception is shown in map view on figure 3. Lambert (18) reports the presence of several vertical fractures that may have been propagated (or invaded) at slightly different times during the drilling and completion of the borehole. All fractures in coal were propagated within face cleats in a narrow zone (less than 1 in wide) with an average azimuth of N 65° E. Naturally occurring roof joints in the area had essentially the same orientations as the coal face cleat.

One fracture set extending in two wings from the borehole contained drilling mud and cuttings in a vertical face cleat plane. This fracture was apparently induced during the drilling of the short hole below the casing. The fracture extended from the base to the top of the coalbed and into the overlying roof shale. The fracture in the coalbed was 1/4 to 1/2 in wide, with the widest part of the fracture occurring in the center of the coalbed where the coal was "softest." The thinner part of the fracture was near the roof and floor where the coal was "harder." The maximum lateral extent of this fracture was approximately 5 ft near the top of the coalbed to the northeast of the borehole (fig. 4). The lateral extent of the fracture diminished from the top of the coalbed to the bottom. The extent and geometry of this fracture to the southwest was not reported. Small patches of sand and gel were found in the fracture, indicating partial penetration by the stimulation treatment. The presence of "unbroken" gel in the fractures observed underground 7 months after the treatment is an indication that the viscosity reducing agent (breaker) did not perform as designed to reduce the gel viscosity within a few hours after the treatment was completed.

A second fracture (or upward extension of the mud-filled fracture) filled with the cement used to set the casing, was observed in the roof and a short distance into the top of the coalbed. The cement-filled fracture in the roof was traced 133 ft to the northeast and 48 ft to the southwest of the borehole and was generally 1/4 in wide. The fracture penetrated 13 in into the coalbed on the rib 24 ft northeast of the borehole and 7 in into the coalbed on the rib 48 ft southwest of the borehole. No penetration of the cement-filled roof fracture into the coalbed was reported at its maximum extent to the northeast. No evidence of horizontal fractures or adverse effects from the roof penetration by the vertical fractures was reported.

It was speculated by Lambert (18) that the bulk of the treatment was placed into the floor rock exposed in the

wellbore below the coalbed. The primary evidence for this is the minimal amount of treatment fluids and sand observed in the exposed coal section around the borehole. Also, gas was bubbling up through the water filling the open wellbore in the mine floor. It was further speculated that the lower gas production rates observed from this borehole were the result of fractures being filled with drilling mud, cement, and unbroken gel, which prevented propagation of sand-filled fractures in the coalbed and also reduced fracture permeability near the wellbore, thus hindering gas and water flow.

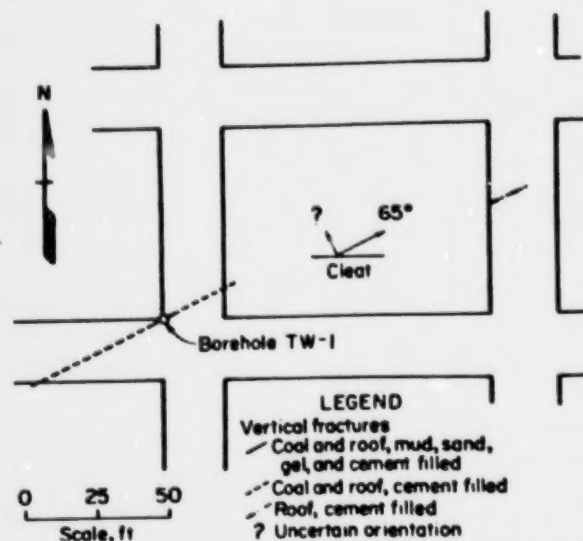


Figure 3.—Mine map, borehole TW-1, with fracture orientations (18-19).

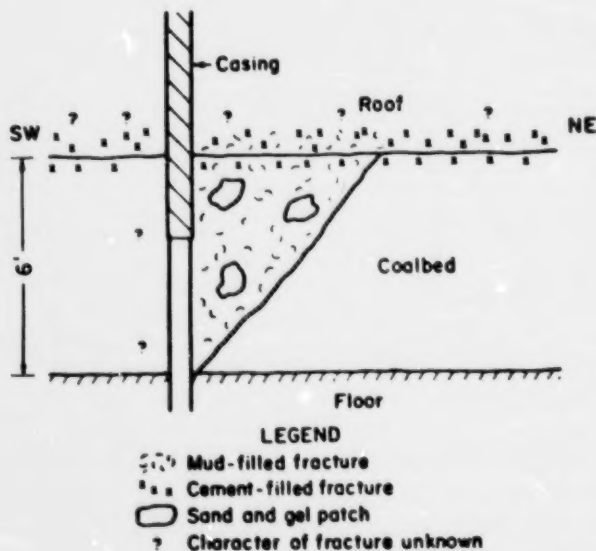


Figure 4.—Schematic cross section parallel to face cleat fractures at borehole TW-1 (18).

OAK GROVE MINE, BOREHOLE TW-2 (20)

Borehole Completion

Borehole TW-2 was drilled 6 1/4 in. in diameter to a depth of 857 ft, approximately 235 ft above the target Blue Creek Coalbed. The remainder of the hole, to the total depth of 1,150.6 ft, was drilled 6 1/4 in. in diameter using an air-foam drilling fluid. The borehole was cased to its total depth with 4 1/2-in.-OD casing and the bottom 500 ft was cemented with 13.8-lb gal cement. The top of the Blue Creek Coalbed was at a depth of 1,093.4 ft and the coalbed was 5.0 ft thick. At the time of completion, borehole TW-2 was located 600 ft in advance of mining.

Treatment Implementation (October 24, 1976)

Prestimulation strata characterization tests included natural gamma ray and gamma density wireline geophysical logs. Lambert (20) reported several interesting observations related to the use of the density logs, including the identification of the location of the slots cut into the casing and correlation of density changes in the coalbed to the type of fractures observed underground. These observations are summarized in the "Underground Observations" section of this discussion.

A jetting tool was used to cut slots through the casing at the target coalbed. The intent was to cut four vertical slots 90° apart, from the base of the coalbed to within 1 ft of the top. The tool was positioned using wireline geophysical logging tools.

The stimulation treatment record for borehole TW-2 is shown in figure 5. The treatment utilized gelled water as specified in table 1 and sand proppant. The stimulation

design selected would theoretically produce two vertical sand-filled fracture wings, 1 in wide and 5 ft high, each extending approximately 150 ft out from the borehole. The actual treatment was characterized by high injection pressures that averaged 2,400 lbf/in²(ga) at the surface. Maximum pressure recorded was 2,500 lbf/in²(ga). No obvious formation breakdown was observed by Lambert (20). An instantaneous shut-in pressure was recorded at 2,200 lbf/in²(ga) surface pressure. Fluid injection rates averaged approximately 8 bbl/min. Sand concentrations were generally 2 lb/gal, but were briefly increased to 4 lb/gal prior to completing injection of the planned sand weight. A total of 3,500 gal of gelled water and 4,000 lb of sand proppant (1,000 lb of 20/40-mesh, 3,000 lb of 10/20-mesh) were used in the treatment.

During uninterrupted water pumping periods, gas production averaged about 15,000 ft³/d with maximum temporary rates of 80,000 ft³/d immediately after servicing the downhole water pump. A complete history of the production rates and problems can be found in the paper by Lambert (20).

Underground Observations

Borehole TW-2 was mined through on February 15, 1977, approximately 4 months after stimulation. The area of the borehole intercept is shown in map view on figure 6. Figure 7 is a cross-section drawing of the face advanced approximately 4 ft beyond the borehole showing the induced fractures and the exposed well casing. Observations of the cased borehole revealed a poor cement job, with only about half of the annular space between the casing and the formation filled with cement in the roof and floor. Sand and/or gel filled the remaining portion of the annular space. No cement was found adhering to the casing within the coalbed interval. The only fully penetrating slots through the casing were found in the floor rock interval below the target coalbed. The casing had expanded outward in this interval where it was pierced by four roughly diamond-shaped slots nearly 5' in long and 2 to 2 1/2 in wide. The tops of the slots were

Table 1.—Specifications of gelled stimulation fluid, borehole TW-2 (20)

	Base fluid	Conc. per 10 ³ gal H ₂ O
Surfactant	gal	3
Fluid loss additive	lb	50
Gelling agent	lb	66.7
Breaker	lb	2
Complexer	gal	0.4

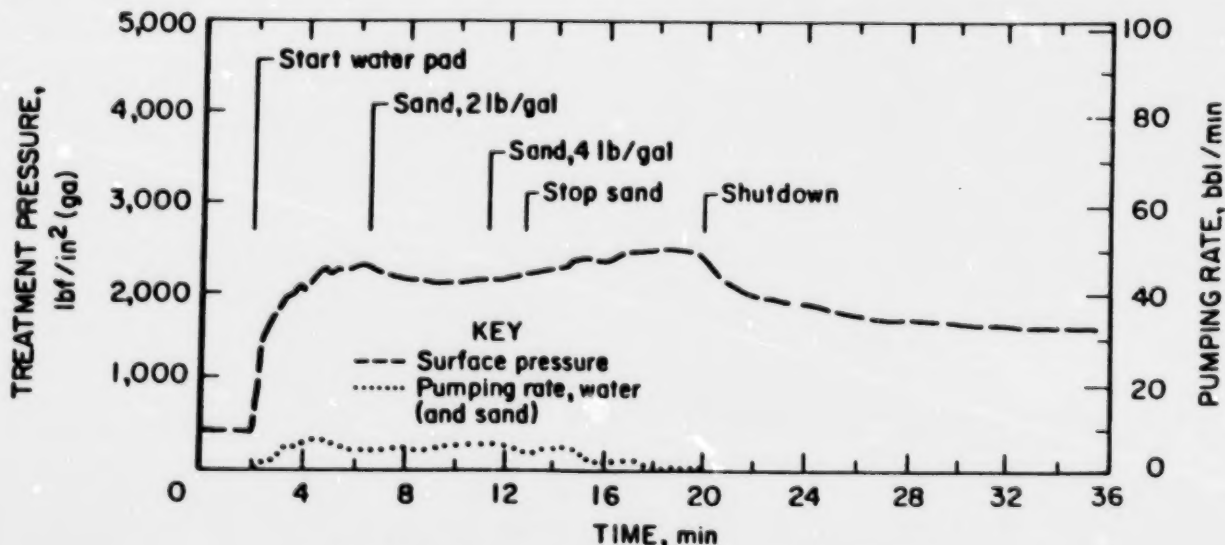


Figure 5.—Treatment record, borehole TW-2. (Modified from Lambert (20))

about 4 in below the base of the coalbed and were oriented N 60° E and N 15° W. In addition to the slots, five holes (0.3-in maximum diameter) were located in an interval 5 to 9 in above the slots.

Lambert (20) reported that before the start of production the presence of the slots only in the floor rock below the target coalbed could have been revealed by a comparison of density logs from before and after slotting operations (fig. 8). The logs run after the slotting show a significant density decrease along the casing interval identified by underground observation to have been successfully slotted.

Factors reported to have possibly contributed to the poor slotting included insufficient time allowed to cut through the casing; slight rotation or twisting of the tool

as it was lowered through the prescribed interval; and the very soft, friable nature of the coal, which may have caused absorption of much of the cutting energy. Underground observations supporting these hypotheses include the presence of vertically oriented grooves on the inner surface of the casing opposite the coalbed interval where the casing had not been completely cut through, and location of the successfully cut slots and perforations adjacent only to the "hard" floor rock and "hard" lower portion of the coalbed.

The location of the slots in the hard floor rock below the target coalbed did not prevent the stimulation fluids from entering the coalbed, but may have contributed to the high initial treatment pressures. The high pressures throughout the treatment were attributed by Lambert (20) to the high viscosity of the treating fluid and to the flow resistance created by the small openings in the casing.

Three well-defined vertical sand-filled fractures were present in the hard lower coal interval shown in figure 7. The fracture on the right side of the face (location 1, figure 7) was 4½ in wide near the floor. To date this is the widest sand-filled coalbed fracture observed underground. The fracture narrowed significantly at the interface between the lower hard coal and the upper soft friable coal, inclined and eventually became horizontal near the top of the coalbed. The other two vertical fractures in the lower coal section (locations 2 and 3, figure 7) inclined and became horizontal at or near the hard-soft coal interface. Each of these fractures terminated at the base of the coalbed and did not penetrate the roof rock.

The vertical portions of the three sand-filled fractures were oriented as shown on figure 6, and, according to Lambert (20), "followed local fracture planes in the coal." The fractures to the north of the borehole (locations 2 and

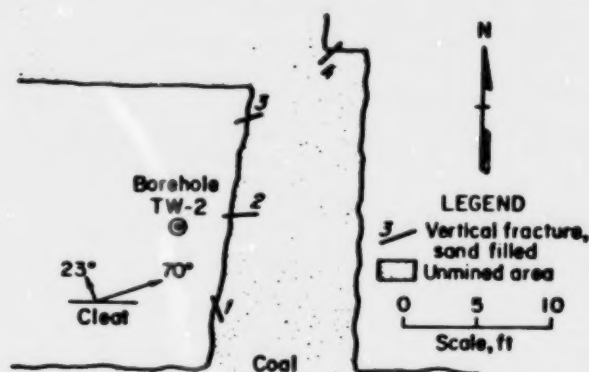


Figure 6.—Mine map, borehole TW-2, with fracture orientations. (Modified from Lambert (20))

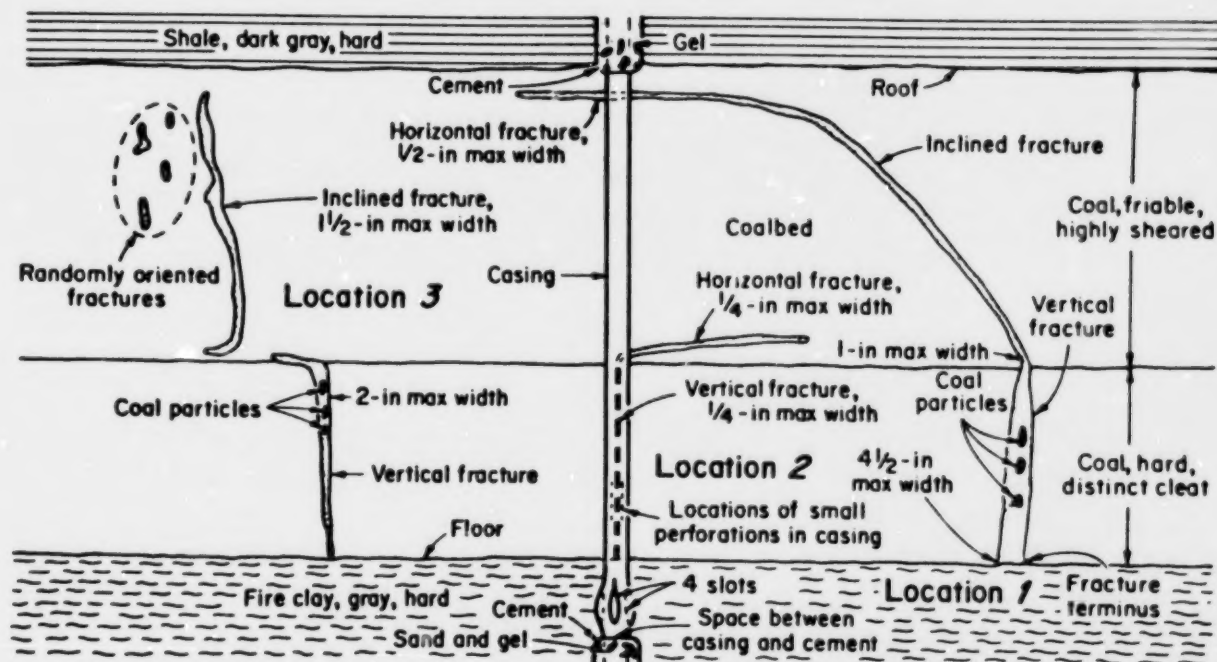


Figure 7.—Cross section of coal face at borehole TW-2, with fracture orientations. (Modified from Lambert (20))

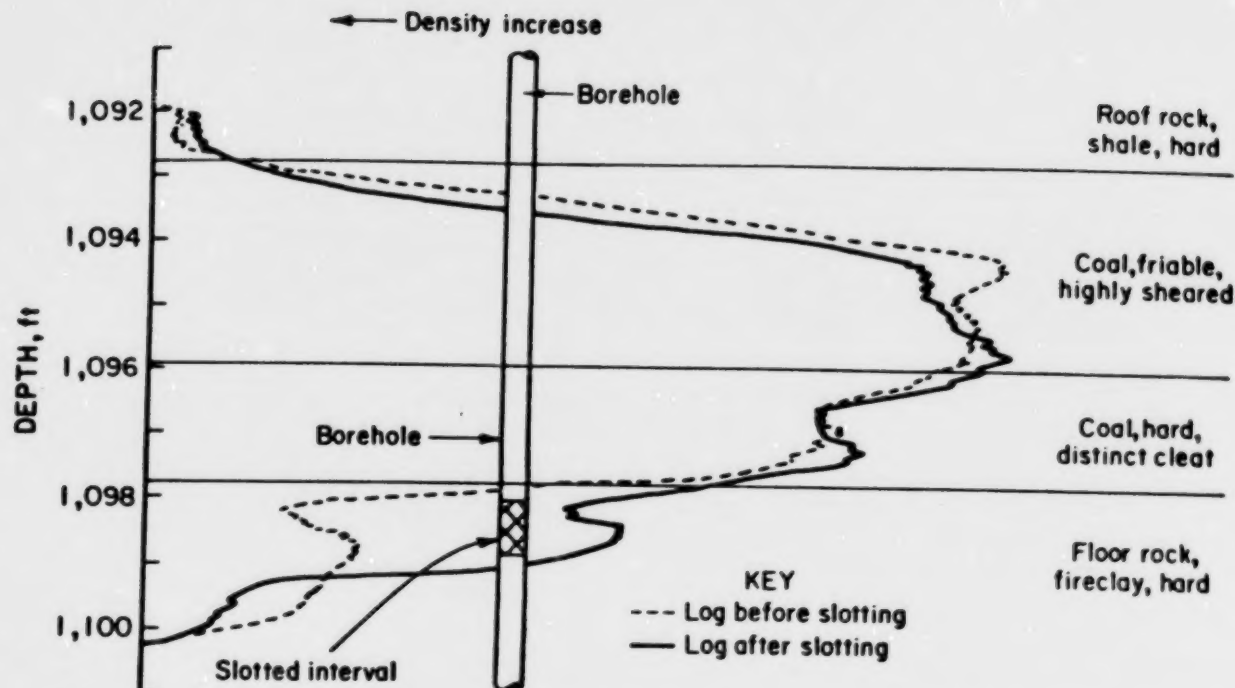


Figure 8.—Comparison of gamma density logs before and after jet slotting, borehole TW-2. (Modified from Lambert (20))

3, figure 6) were oriented generally in the face cleat direction (N 70° E), and those to the south (location 1, figure 6) appeared to be developed along butt cleats (N 23° W).

The two primary fractures observed in the more friable upper section of the coalbed appeared to have intercepted and propagated along inclined shear planes in the coal. No penetration of the roof strata was observed.

In addition to the common orthogonal vertical cleat system found in most coalbeds, the coal section at the Oak Grove Mine and other mines in the area displays secondary vertical and inclined fracture systems that intersect at various angles and with a variety of orientations (16). These complicated fracture systems are the result of the complex geological history of the area, and they can apparently influence the character of the induced fractures. Lambert (20) pointed out that the vertical fractures were contained in the lower, hard, distinctly cleated coal zone (correlated to the denser interval on the density log, figure 8) and that horizontal and inclined fractures were contained in the upper, friable, highly sheared portion of the coalbed (correlatable to less dense interval on the density log).

Examination of the entries developed around borehole TW-2 revealed only one other exposure of sand-filled vertical fractures (location 4, figure 6) to the northeast of the borehole. The maximum lateral extent of vertical sand-filled fractures was about 16 ft from the borehole. Horizontal sand-filled fractures extended a maximum of 8 ft from the borehole. It should be pointed out that the maximum 16 ft of observed lateral propagation of the vertical fractures is far short of the designed 150-ft sand-filled fracture length. The gelled fluids used in the

treatment included an enzyme-type breaker to reduce the fluid viscosity within a few days, but the presence of unbroken gel around the casing as observed underground 4 months after the treatment, as previously described in the section on borehole TW-1, indicates that the breaker did not perform as intended. If unbroken gel was still present in the induced fractures, it is quite likely that permeability and gas production would be reduced.

OAK GROVE MINE, BOREHOLE TW-3 (21)

Borehole Completion

Borehole TW-3 was drilled 8 in. in diameter to a depth of 1,001.2 ft, approximately 73 ft above the top of the prime target of the Blue Creek Coalbed. The remainder of the hole was drilled 7 1/4 in. in diameter, to the total depth of 1,113.5 ft, using foam as the drilling fluid. The borehole was drilled to a depth 30 ft below the base of the 5-ft-thick Blue Creek Coalbed. The borehole was cased with 4 1/2-in.-OD casing to a depth of 1,066 ft, approximately 2 ft into the 3-ft-thick Mary Lee Coalbed. Approximately 7 ft of rock separated the Mary Lee and Blue Creek Coalbeds.

The casing was cemented in place using a cement packer shoe, casing centralizers, and cement baskets in an effort to keep the cement from reaching the Blue Creek Coalbed in the open hole below the bottom of casing. The risk associated with this procedure was of cement leakage around the packer shoe and the potential of formation damage in the coalbed. A latex-based, low-fluid-loss class H cement with a 15.3-lb/gal weight was used in borehole TW-3. At the time of completion, the borehole was approximately 590 ft ahead of mining.

Treatment Implementation (November 21, 1976—Blue Creek, July 21, 1977—Mary Lee)

Prestimulation strata characterization tests included natural gamma ray, density, resistivity, self-potential, and neutron wireline geophysical logs. A jet slotting tool was used to cut three vertical slots, spaced 120° apart, into the Blue Creek Coalbed. This procedure was utilized to cut through any formation damage in an effort to ensure an open pathway into the coalbed prior to the stimulation treatment.

The Blue Creek Coalbed was stimulated with a 75-pct-quality nitrogen-generated foam and sand proppant. The stimulation treatment record for this borehole is shown in figure 9. An initial formation break was interpreted by Lambert (21) to have occurred at a surface pressure of 700 to 750 lbf/in²(ga). After the initial formation break, fluid injection was briefly stopped to determine the instantaneous shut-in pressure, which was interpreted to be approximately 400 lbf/in²(ga). Maximum and average surface treating pressures were approximately 1,500 and 1,400 lbf/in²(ga), respectively, at injection rates averaging 10 bbl/min. The borehole was stimulated through 2½-in-diameter tubing with a packer set in the casing near the bottom of the hole. A total of 20,000 gal of foam and 25,000 lb of sand (10,000 lb of 80/100-mesh and 15,000 lb of 20/40-mesh) were injected. The finer 80/100-mesh sand was used as a fluid loss additive to allow more of the stimulation fluids to propagate fractures instead of leaking away nonproductively into the formation.

Approximately 7 months after the stimulation of the Blue Creek Coalbed, the overlying Mary Lee Coalbed was stimulated. The Mary Lee Coalbed was characterized by Lambert (21) as being thin (3 ft thick) and containing several shale partings. The exact depth of the Mary Lee Coalbed in the borehole was determined through the use of wireline geophysical logs made prior to perforating the casing with shaped explosive charges.

The treatment was conducted through tubing using 5,040 gal of 75-pct-quality nitrogen-generated foam, 3,350 lb of 80/100-mesh sand, and 3,550 lb of 20/40-mesh size glass beads. The glass beads were used to aid in identifying any penetration of the stimulation treatment

from the Mary Lee Coalbed into the Blue Creek Coalbed below.

A formation break was reported by Lambert (21) to be at a surface pressure of 600 lbf/in²(ga). The maximum surface treatment pressure was reported to be 2,000 lbf/in²(ga) with an average surface treatment pressure of 1,000 lbf/in²(ga) at an average fluid injection rate of 5 bbl/min.

Gas production rates were generally erratic, especially early in the production history, but generally declined from a high of over 140,000 ft³/d to 50,000 ft³/d and then increased to 85,000 ft³/d. After the stimulation of the Mary Lee Coalbed, gas production rose to a maximum of 45,000 ft³/d. A complete production history of this borehole can be found in reference 21.

Underground Observations

A fracture from borehole TW-3 was intercepted in the Blue Creek Coalbed on October 6, 1977, approximately 10½ months after stimulation. The initial interception and underground observation of the fracture from borehole TW-3 did not include the mine-through of the borehole itself. Subsequently, entries were developed to the north of the borehole revealing the borehole and an extension of the earlier observed fractures (fig. 10). The fracture seen near the borehole was described by Lambert (21) as a single vertical sand-filled fracture, beginning as a hairline crack about 3 ft above the base of the coalbed, and gradually widening to approximately ¼ in at the top of the coalbed. The total observed lateral extent of this fracture to the southwest of the borehole was 80 ft.

The fracture continued upward at the same ¼-in thickness for an undetermined distance into the roof. The fractures observed within the coalbed contained mostly 80/100-mesh sand and the sand in the roof fracture was predominantly 20/40-mesh size. The glass beads used to stimulate the overlying Mary Lee Coalbed were not found in the Blue Creek Coalbed. Failure to contain the casing cement above the Blue Creek Coalbed was evidenced by the presence of cement in the same vertical fracture (coal and roof) as the sand proppant and as far as 80 ft from the borehole to the southwest. The orientation of the vertical sand-filled fractures in the coalbed and overlying strata

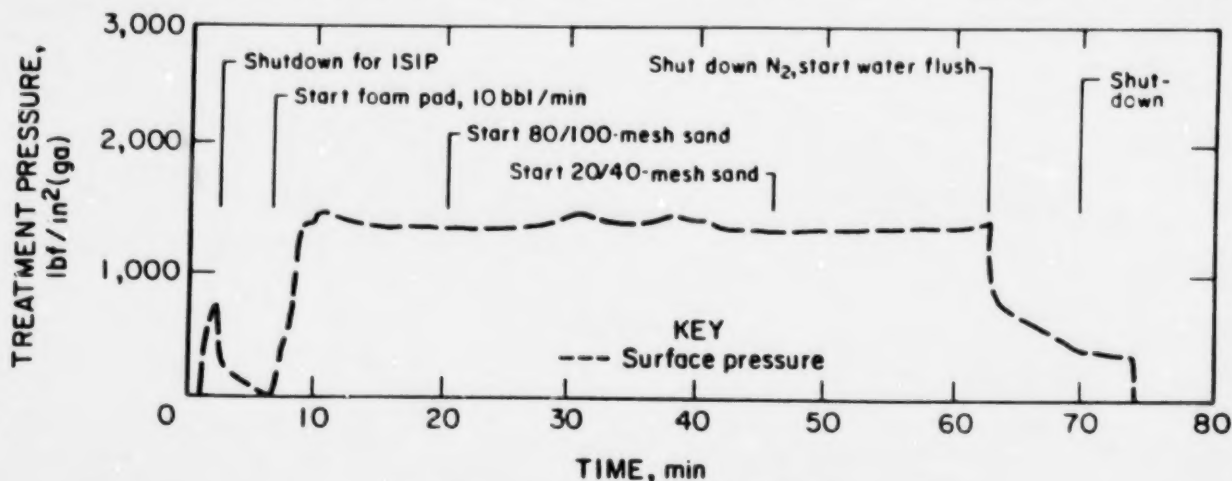


Figure 9.—Treatment record, borehole TW-3 (18).

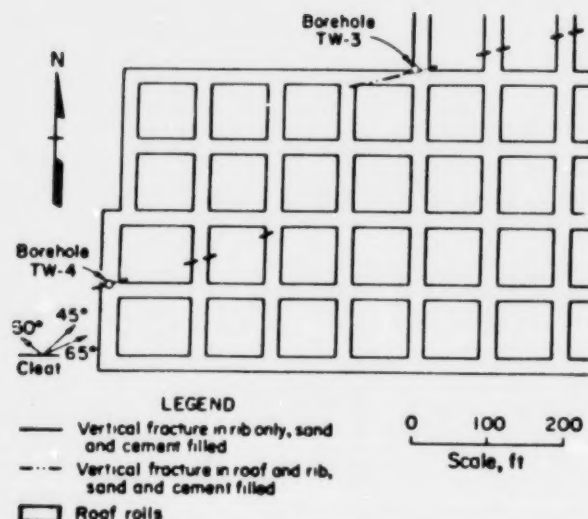


Figure 10.—Mine map, boreholes TW-3 and TW-4, with fracture orientations. (Modified from Lambert (21))

was N 67° E. Compilation of orientation measurements of naturally occurring roof joints (total of 57) in the vicinity of the borehole TW-3 (and the nearby borehole TW-4, to be discussed in the next section) indicated an average roof joint orientation of N 66° E. This orientation is essentially the same as that observed for the sand-filled roof fracture near borehole TW-3, and may be an indication that the sand-filled roof fracture was a preexisting roof joint entered by the stimulation treatment. The analysis of 357 cleat orientations in the same area produced a relatively broad range of readings (most face cleats were between N 45° E and N 65° E) with an average of N 53° E reported by Lambert (21). No horizontal fracture component was observed.

Borehole TW-3 was eventually mined through and a sand-filled vertical fracture trending N 68° E was traced across three new entries for a total lateral extent of 210 ft. Details of this fracture wing exposure are not available.

Supplementary roof support was installed in the vicinity of borehole TW-3. This is the only reported case out of 22 Government-sponsored stimulation treatments where supplementary support was deemed necessary by mine management. The evaluation of the situation and description of the additional support was described by mine management (21) as follows:

The roof in the general area of TW-3 was of a quality typical of most of the mine developed to date; however, in the immediate area of the bottom of TW-3 in which both the upper and lower coal seams had been fractured, it was necessary to install supplementary roof support. This supplemental roof support was required when mine management observed roof movement along the west rib of the entry driven due north of the bottom of the borehole, additional draw rock separation, roof becoming excessively drummy, and water seeping out of cracks in the roof. The supplemental roof support consisted of boxing the intersection immediately to the south of the borehole with 6 inch H-beams, installing 4-inch H-beams supported by timbers for

distances of 70 ft to the east and 40 ft to the north of the borehole, and installing 10-foot-long expansion-shell anchored roof bolts between the standard 4-foot-long resin roof bolts to ensure that the bolts anchored in the solid rock above the upper coal seam.

Installation of the supplementary supports required additional time as well as close inspection by mine management to ensure its adequacy; however, it was possible to mine through this area without experiencing any roof fall during mining operations.

Lambert's (21) comments on the roof conditions in this area are as follows:

The entry nearest TW-3 is also shown (fig. 10) to be a location of natural roof disturbances. Geologists also noted the presence of many wet roof joints in this area. Except for the single sand-filled crack leading from TW-3 (fig. 10), no evidence could be found to indicate that these wet roof joints had been created by stimulation. Unusually large amounts of seepage from roof openings near TW-3 could, however, be due to the borehole's presence, since it is an accumulation point for water within a coal unit only 7 ft above the open entry.

Lambert (21) further commented on possible reasons why the stimulation treatment for borehole TW-3 (and TW-4, to be discussed in the next section) did not remain in the coalbed:

1. The fine-mesh sand used as a fluid-loss agent was the only sand found to be contained in the coal. This sand is thought to have caused the propagating fracture to clog or screen-out in the coal early in the treatment. With continued pumping, it is possible that the fracturing fluids entered a preexisting plane of weakness in the shale roof rock. It is unlikely that this very hard roof rock was actually "broken" by the 1,400 psig pressure used during stimulation. No evidence could be found at either underground test site to indicate that the roof offered much resistance to the stimulation. Such evidence would include the presence of horizontal fractures or short propagations along butt cleat directions as was seen in other stimulation treatments.

2. Cement used to set casing in TW-3 (and TW-4) was found in the roof, and up to ≈ 80 ft away from the wellbores. The roof strata was already broken or fractured sometime before casing was cemented in the test wells, probably just after the time when the holes were drilled.

OAK GROVE MINE, BOREHOLE TW-4 (21)

Borehole Completion

Borehole TW-4 was drilled 510 ft southwest of borehole TW-3 (fig. 10). It was drilled 8 in. in diameter to a depth of 995.5 ft, approximately 70 ft above the top of the target Blue Creek Coalbed. The remainder of the hole was drilled to a 7½-in diameter to the total depth of 1,086 ft, using clear water as the drilling fluid. The borehole was drilled to a depth 15 ft below the base of the 5-ft-thick Blue Creek Coalbed. The borehole was cased with 4½-in-OD casing to a depth of 1,065.7 ft, approximately 0.7 ft into the top of the Blue Creek Coalbed. The casing was

cemented in place using a cement packer shoe and associated equipment and cement specifications as used on borehole TW-3. At the time of completion, the borehole was approximately 1,010 ft ahead of mining.

Treatment Implementation (July 18, 1977)

Prestimulation strata characterization tests included gamma ray, density, resistivity, and self-potential wireline geophysical logs. A jetting tool was used to cut two slots, spaced 180° apart, into the coalbed.

The Blue Creek Coalbed was to be stimulated through 2½-in-diameter tubing with a 75-pct-quality nitrogen-generated foam with sand proppant similar to that of borehole TW-3. However, the stimulation treatment had to be terminated prematurely when foam containing sand was observed flowing from around the outside of the casing at the wellhead, indicating a poor cement job. A total of 12,200 gal of foam with 12,520 lb of sand (10,000 lb of 80/100-mesh and 2,520 lb of 20/40-mesh) was used in the treatment. An initial formation break was interpreted by Lambert (21) to have occurred at a surface pressure of 700 to 750 lbf/in²(ga), similar to that observed at borehole TW-3. The maximum surface treating pressure was 1,800 lbf/in²(ga), with an average surface pressure of about 1,500 lbf/in²(ga) for the treatment at injection rates averaging 10 bbl/min. The continuous stimulation treatment record for this borehole is shown in figure 11.

Borehole TW-4 was not equipped with a water pumping system, however, it still produced gas at rates as high as 183,000 ft³/d. Rates averaged 150,000 ft³/d then dropped sharply to the final measured rate of 40,000 ft³/d prior to plugging. Lambert (21) reported that there was some indication when comparing the production histories of boreholes TW-3 and TW-4 that the two wells interfered with each other. It was also reported that there was evidence of nitrogen gas communication to TW-3 when TW-4 was stimulated, indicating the movement of fracture fluids to greater distances than could be traced by the presence of sand-filled fractures underground.

Underground Observations

Borehole TW-4 was mined through on November 16, 1977, approximately 4 months after stimulation. A single

vertical sand-filled (mostly 80/100-mesh size) fracture, similar to those observed at the TW-3 borehole location, was found in the vicinity of borehole TW-4 (fig. 10). It began as a hairline crack about 3 ft above the base of the Blue Creek Coalbed, and gradually widened to approximately ⅜ in at the top of the coalbed. The maximum lateral extent of this fracture to the northeast was about 220 ft on an average azimuth of N 67° E, similar to the orientation of naturally occurring roof joints in the area. Sand was again found in a roof fracture above the coalbed as at borehole TW-3. Mining did not proceed more than a few feet to the west of the borehole, therefore the maximum lateral extent in this direction is unknown.

A large quantity of casing cement was found below the packer shoe at borehole TW-4. Cement covered most of the exposed wellbore in the coalbed, except where the two slots, each 5 in wide, were cut 9 to 12 in into the coal. The orientation of these slots was found to be approximately 90° to the orientation of the sand-filled vertical fracture. Cement was also found in the sand-filled vertical fracture in the coal and roof.

An evaluation by mine management (21) of the roof conditions in the vicinity of borehole TW-4 is as follows:

Because the roof of the mine in the general vicinity of TW-4 was of a quality which required installation of more roof support than used in many other portions of the mine, it was difficult to assess the effect of the fracture on the mine roof at this location. However, no further increment of supplemental roof support was required at the base of the borehole or along the fracture wings.

Lambert's (21) comments on the roof conditions in this area are as follows:

Previous geologic inspection of the mine near TW-4 showed the well to be nearly surrounded by minor deformation and dislocations in the coal associated with lenticular shale ridges projecting from the roof. Areas of the mine where these features, called "rolls", were observed are shown in figure 10. Because inherent rock weaknesses develop near such features, these areas generally require some degree of additional roof support. The necessity for more roof support near TW-4 than used

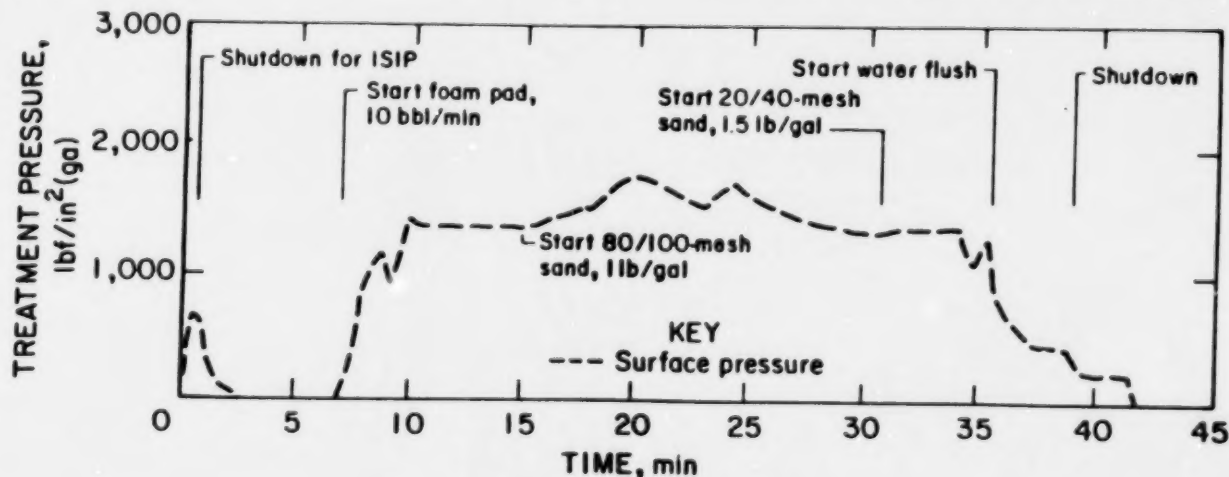


Figure 11.—Treatment record, borehole TW-4 (18).

in many other portions of the mine, as reported by mine management, may be attributed to this geologic phenomenon rather than stimulation of the test well. The fact that additional roof support was not required at the base of TW-4 nor was it required along the fracture wings indicates borehole stimulation was not the cause for providing additional roof support.

A discussion of the possible reasons why the stimulation treatment penetrated the roof is given in the previous section on borehole TW-3.

OAK GROVE MINE, BOREHOLE TW-5 (9, 22)

Borehole Completion

Borehole TW-5 was drilled 8 in. in diameter to a depth of 1,103.3 ft and 6½-in.-OD casing was installed to a depth of 1,102 ft, approximately 43 ft above the top of the Blue Creek Coalbed at 1,145 ft. A 5½-in.-diameter hole was then drilled out from the bottom of casing using air as the drilling fluid. This open hole section extended through the Mary Lee and Blue Creek Coalbeds to a total depth of 1,158.9 ft, approximately 8 ft below the base of the Blue Creek Coalbed. A stratigraphic section of the coal interval is shown in figure 12. The Mary Lee Coalbed is 2.5 ft thick and is separated from the 5.5-ft-thick Blue Creek Coalbed below by 6 ft of shale and thin coal stringers. The Blue Creek Coalbed was the target for gas production.

Treatment Implementation (December 16, 1978)

Prestimulation strata characterization included a suite of wireline geophysical logs (caliper, natural gamma ray, and density) and direct observation of the open hole

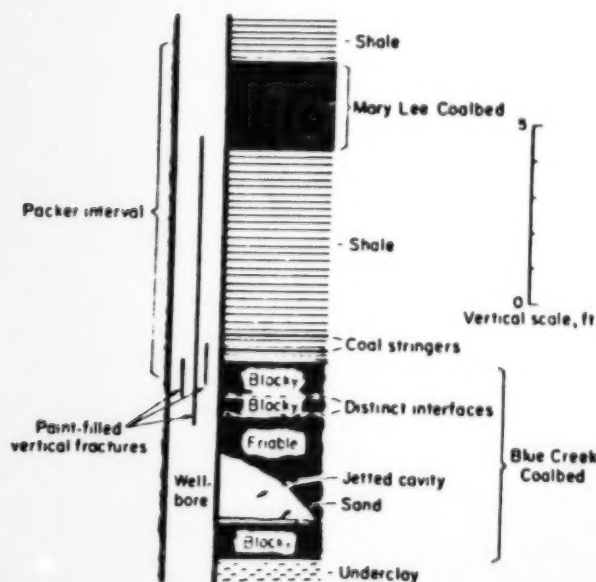


Figure 12.—Stratigraphic section and near-wellbore underground observations in the Mary Lee-Blue Creek Coalbed intervals, borehole TW-5. (Modified from Lambert (9) and Mahoney (22))

section with a submersible borehole TV camera. The TV camera was used to make a video recording of the borehole conditions for comparison with a planned poststimulation survey. The initial survey (22) indicated that the noncoal strata were "tight and unjointed" and the two coalbeds were "somewhat porous and cleated, although very little spalling had occurred during drilling."

Prior to stimulation, a horizontal slot was cut into the face of the Blue Creek Coalbed using a jetting tool. The two-port tool (180° apart) was positioned approximately at the midpoint of the coalbed and was continuously rotated at approximately 4 rpm for 27 min. A total of 2,268 gal of water and 5,500 lb of sand was used in the slotting process at a rate of 2 to 2.75 bbl/min. An average wellhead pressure of 1,800 lbf/in²(ga) was observed during the slotting procedure. Coal returns were observed in the fluid returning to the surface. A downhole TV survey confirmed that a full 360° cavity had been eroded in the target coalbed.

To isolate the Blue Creek Coalbed from the rest of the open-hole section, an inflatable 10-ft-long packer was run on the bottom of the tubing. The rubber packer element was placed as shown in figure 12, and extended from 0.5 ft below the top of the Blue Creek Coalbed to a point 0.5 ft above the top of the Mary Lee Coalbed. Initial breakdown was achieved using clear water at a maximum pumping rate of 1.25 bbl/min. The breakdown pressure measured at the wellhead was approximately 1,000 lbf/in²(ga). Before the primary foam treatment could be started, water was observed flowing at the wellhead from the annular space between the casing and the tubing. This observation was an indication that a fracture may have propagated through the strata around the packer, which would interfere with injection of an effective treatment into the Blue Creek Coalbed. The treatment was terminated at this point and the packer was removed from the hole. A slug (10 bbl) of fresh water containing 3 gal of red fluorescent paint and 1 lb of fluorescein dye, was injected into the formation for future underground characterization followed by an additional 10 bbl of water. A total of 2,200 gal of water was injected into the formation prior to abandonment of the initial treatment attempt.

Examination of the packer after removal revealed two 180° opposed, ¼-in.-wide impressions along a 6-ft length of the rubber element. These impressions were further characterized (22) by use of the TV camera as "score marks" in the wellbore "extending in a vertical direction from the top of the Blue Creek seam through the middleman to the base of Mary Lee seam." Mahoney (22) indicated that it was possible that the pressure used to inflate the packer element was sufficient to create the observed cracks before the actual injection of the water.

The packer was reinstalled at its previous position in the wellbore, and the Blue Creek Coalbed was successfully treated without any apparent leakage of treatment fluids around the packer. The treatment for borehole TW-5 was designed to utilize a 75-pct-quality nitrogen-generated foam without any sand proppant to investigate the production characteristics and the effect on the surrounding strata resulting from a no-proppant stimulation. The benefit expected from a no-proppant stimulation was confinement of the treatment to the coalbed (minimizing the potential for roof penetration and possible roof damage) by using low injection rates (9). Cost savings from deleting the sand proppant from the stimulation treatment were also anticipated, both from the reduced

initial expenditures and also by decreasing the workover costs related to downhole pump failures experienced early in the production phase on previous boreholes using sand proppant.

Table 2 summarizes the treatment implemented in borehole TW-5. The continuous stimulation treatment record for this borehole is shown in figure 13. Foam injection began at a rate of 2 bbl/min and was increased in 1-bbl/min increments at 30-min intervals up to a rate of 5 bbl/min. The 5-bbl/min rate was maintained for 1.5 h and was then increased to 6 bbl/min for an additional 1.5 h. A total of 53,000 gal of 75-pct-quality nitrogen-generated foam was injected into the formation. Blue fluorescent paint and fluorescein dye were also injected with the foam to differentiate movement of the foam treatment fluid from the initial water injection when the treatment area was eventually intercepted by mining. The surface pressure gradually increased from about 1,500 lb/in²(ga) to a maximum of 2,000 lb/in²(ga) as the injection rate increased from 2 to 6 bbl/min. Average surface treatment pressures were about 1,900 lb/in²(ga).

A total of 6.6 MMft³ of gas at an average rate of 49 Mft³/d was produced during the 4.5-month production life of the borehole. A complete production history can be found in reference 22.

Underground Observations

Borehole TW-5 was mined through on July 12, 1979, approximately 7 months after stimulation. The borehole

Table 2.—Summary of treatment chronology, borehole TW-5 (22)

Time, min	Foam		Wellhead pressure, lb/in ² (ga)
	Injection rate bbl/min	Volume, bbl	
0 to 30	2	60	0-1,550
30 to 60	3	90	1,600
60 to 90	4	120	1,700
90 to 180	5	450	1,750
180 to 270	6	540	1,750-2,000

was intercepted very close to the southeast corner of a pillar (fig. 14). A large cavity eroded by the jet slotting procedure was observed near the base of the borehole. The cavity appeared to be conical in shape with its base approximately 1 ft above the bottom of the Blue Creek Coalbed (fig. 12). Sand from the slotting procedures covered the bottom of the cavity. The height of the cavity was 23 in and the radius of the base was approximately 36 in.

Examination of the exposed wellbore revealed the two 180° opposed cracks first observed in the packer element and with the TV camera. The two primary fractures were developed in the upper part of the Blue Creek Coalbed and extended through the middleman into the base of the overlying Mary Lee Coalbed. Two additional short vertical fractures were also present near the top of the Blue Creek Coalbed (fig. 12). A paint-coated hairline vertical crack extended approximately 8 ft from the borehole and, according to Lambert (9), appeared to be a lateral extension of the vertical fracture observed at the wellbore. This crack was observed in the roof strata above the commonly mined interval. The commonly mined height includes the two coal stringers and intervening carbonaceous shale directly above the main bench of the Blue Creek Coalbed. The roof crack was oriented at N 66° E, close to the N 61° E predominant face cleat orientation.

The only other evidence of penetration of fracture fluids into strata above the mined interval was the presence of blue fluorescent paint in roof joints approximately parallel to the face cleat orientation at locations 25, 29, 30, and 31 at the distal end of the fluid penetration to the southwest (fig. 14). No adverse mining conditions or roof movement were reported associated with these roof penetrations.

Figure 14 shows the location and orientation of vertical paint-filled fractures observed in the vicinity of borehole TW-5. If an orientation on the fracture was not obtained but was recorded as being in the face cleat direction, the fracture trace was plotted as being oriented at an azimuth similar to that measured for nearby fractures. These fractures are identified by a ? in figure 14.

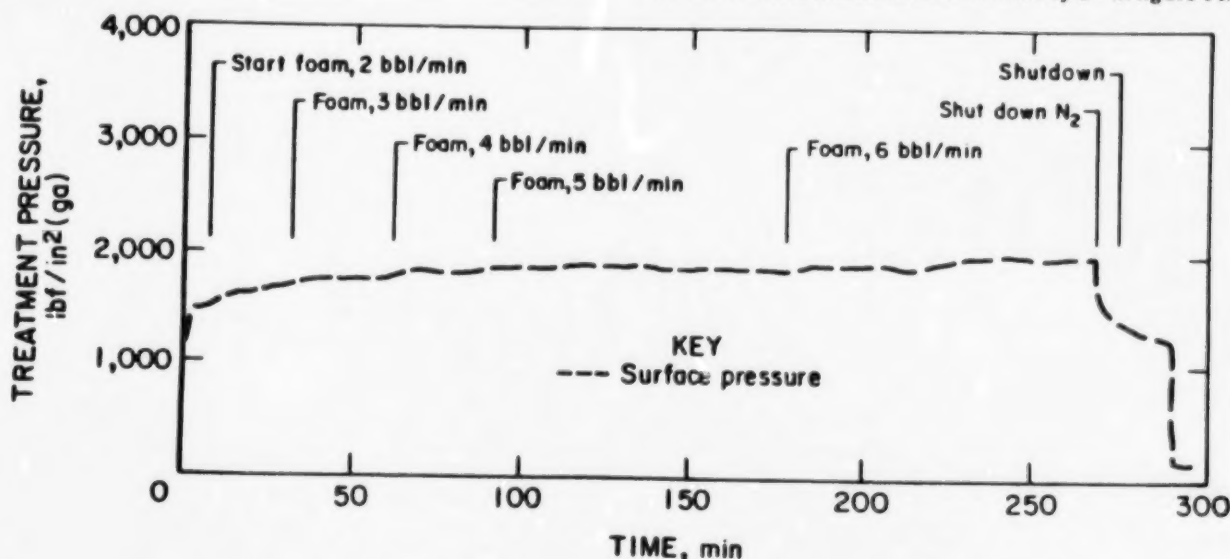


Figure 13.—Treatment record, borehole TW-5 (19).

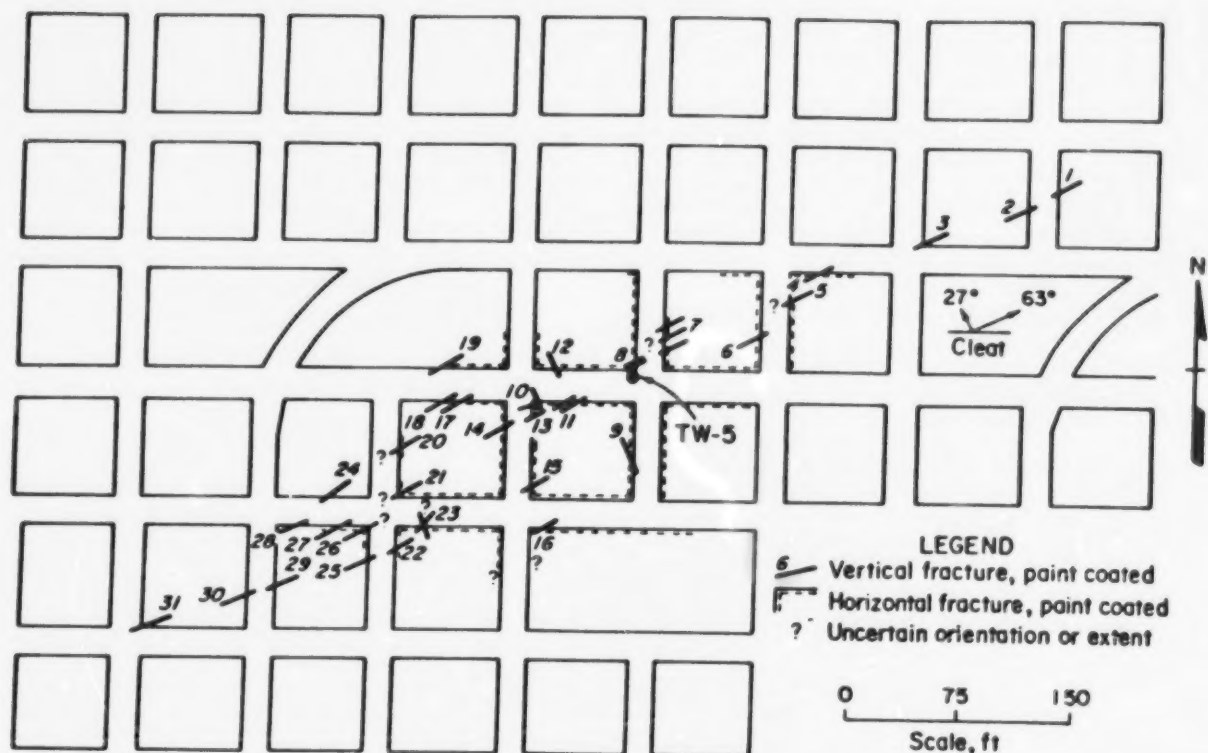


Figure 14.—Line map, borehole TW-5, with fracture orientations. (Modified from Mahoney (22) and Boyer (19))

Table 3.—Summary of fracture observations (paint-coated surfaces), borehole TW-5 (19)

Location ¹	Fractures	Zone width, ft	Fracture description, vertical	Orientation	Paint color	Fracture description, horizontal
1	1	NAP	Top of BC to 1st coal stringer	N 60° E	NR	NR
2	1	NAP	Top of BC to 2d coal stringer	N 63° E	NR	NR
3	1	NAP	Top 3 in of BC to 2d coal stringer	N 62° E	NR	NR
4	3	1	Top 1 in of BC to 2d coal stringer	N 61° E	NR	NR
5	1	NAP	NR	NR	Blue	NR
6	1	NAP	NR	N 61° E	Red	NR
7	Zone	20	Small	NR	NR	NR
8	Zone	5	NR	N 63° E	Red, blue	NR
9	1	NAP	NR	N 65° E		
10	Zone	NR	Face cleat—top 15 in of BC to 1st coal stringer, butt cleat—generally top 8 to 15 in of BC, max top 30 in of BC	N 29° W N 69° E N 26° W	NR NR	NR NR
11	Zone	8	Floor to roof	N 54° E N 59° E	1 red, rest NR	On top of 1st coal stringer
12	1	NAP	Top 18 in of BC to 1st coal stringer	Butt Cleat	NR	NR
13	Zone	2	Floor to roof	N 60° E	NR	On roof and on interface between blocky zones in BC
14	Zone	5	Several floor-to-2d-coal stringers, rest NR	N 59° E	NR	On top of 1st coal stringer
15	Zone	8	NR	N 58° E	NR	NR
16	1	NAP	Top 30 in of BC	N 60° E	NR	NR
17	1	NAP	Floor to 1st coal stringer	NR	Blue	On top of 1st coal stringer
18	1	NAP	Top 12 in of BC to below 3d coal stringer	N 59° E	do	NR
19	1	NAP	Top of BC to 1st coal stringer	N 59° E	do	On top of 1st coal stringer
20	Zone	NR	Floor to roof	Random	do	NR
21	1	NAP	do	NR	do	On top of 1st coal stringer
22	2	NAP	do	N 56° E	do	NR
23	2	NAP	Intersect at 60°	NE and NW	do	NR
24	1	NAP	Middle of BC to roof	N 50° E	do	NR
25	1	NAP	Floor to roof	NR	do	NR
26	1	NAP	Floor to 1st coal stringer	NR	do	On top of 1st coal stringer
27	1	NAP	do	N 62° E	do	Top of BC
28	1	NAP	Middle of coal to top of coal	N 67° E	do	NR
29	1	NAP	Middle of coal to 2d coal stringer	N 56° E	do	NR
30	1	NAP	do	N 56° E	do	NR
31	1	NAP	do	N 62° E	do	NR

BC Blue Creek Coalbed. NAP Not applicable. NR Not reported. ¹ Shown on figure 14.

Additional information on the actual observations for each numbered fracture or fracture zone is given in table 3.

The lateral extent of the horizontal paint-coated surfaces are also shown in figure 14. Horizontal paint-coated surfaces included the three highly polished slickensided surfaces at the top of the Blue Creek Coalbed and on top of the two thin coal stringers in the 1-ft interval directly above the Blue Creek Coalbed (fig. 12). One or more of the three horizontal surfaces were coated with fluorescent paint at any particular location, and without the presence of the paint, these interfaces penetrated by the stimulation fluids could not be differentiated from "normal" surfaces.

As discussed previously (22), general practice throughout the Oak Grove Mine is to mine up to the second coal stringer because the "slickensided coal surfaces demonstrated very little adhesion to the surrounding strata." Mahoney (22) also indicated that the inherent horizontal weakness at the top of the mined interval "appeared to have prevented upward growth of the induced fractures," and "it was not uncommon to observe an abrupt termination of the vertically decorated planes wherever they abutted one or the other coal stringer." The horizontal paint-coated surfaces as shown in figure 14 cover an essentially elliptical area with the long axis oriented approximately parallel to the face cleat orientation. The maximum extent of paint on a horizontal surface was approximately 230 ft to the southwest of the borehole.

Numerous vertical fractures or zones of paint-coated cleat surfaces were identified around borehole TW-5. Most of these fractures were oriented in the face cleat direction and only four locations (9-10, 12, 23) had butt cleats coated with paint (fig. 14). As with the horizontal fractures, the vertical fractures were only identifiable with the fluorescent paint and were essentially only the width of the face cleat itself. There appears to be a difference in the character of the paths of stimulation fluid movement observed to the northeast and to the southwest of the borehole. There are fewer vertical fractures or zones of fractures to the northeast than to the southwest (7 vs 24). The fractures to the northeast are generally confined to the upper portion of the Blue Creek Coalbed and the approximately 1 ft of overlying strata containing the two 1-in-thick coal stringers. Even the fractures close to the borehole at locations 7 and 8 are contained near the top of the coal interval. The fractures to the northeast are also contained in a narrow band very close to the axis of the elliptical pattern of the horizontal paint distribution.

The vertical fractures to the southwest (fig. 14), in addition to being more numerous, commonly extended the entire height of the coalbed (locations 11, 13-14, 17, 20-22, 25-27) as far as 220 ft from the borehole (location 27). At the remaining locations to the southwest, the vertical paint-coated fractures were confined to the upper portions of the coalbed up to the two overlying coal stringers. Zones of multiple paint-coated cleat surfaces appeared to be slightly more common to the southwest (locations 11, 13-14, 20, 23) than to the northeast (locations 3, 7-8). Single fractures predominated at locations farthest from the borehole. The vertical fractures to the southwest were found in a band with a maximum width of approximately 130 ft (from location 19 to 16) compared to the very narrow band of vertical fractures found to the northeast. The

maximum lateral extent of vertical fractures was 370 ft at location 31 to the southwest.

The reason for the observed differences in the distribution of vertical paint-coated cleat surfaces may be related to a differential penetration capability along the face and/or butt cleat to the southwest and northeast. If penetration of cleat was easier to the southwest of the borehole, fluids would perhaps initially move along a face cleat in that direction until pressures built up sufficiently at the distal end to force the fracture along butt cleat. This stairstepping along face and butt cleats is a fairly common occurrence as noted throughout this report. Multiple repetitions of this stairstepping could produce the wider band of paint-coated fractures to the southwest of the borehole where the only butt cleat fractures were observed. The narrow band of fractures observed to the northeast of the borehole could have been induced at the same time as those to the southwest or near the end of the treatment when treatment pressures were not sufficiently high to continue fracture propagation to the southwest. There is also a wider dispersion of paint on horizontal surfaces to the southwest, perhaps indicating an easier horizontal penetration perpendicular to the long axis of the treatment fluid distribution in this area. The reason for the supposed easier penetration of cleat southwest of the borehole is unknown.

OAK GROVE MINE, BOREHOLE DHM-5 (17)

Borehole Completion

Borehole DHM-5 was drilled 6½ in. in diameter to a depth of 1,376.5 ft, approximately 7 ft above the top of the target Blue Creek Coalbed at 1,383.5 ft, 4 ft into the parting between the Mary Lee and Blue Creek Coalbeds. The borehole was cased with 4½-in-OD casing to the total depth of the hole and the casing was cemented in place with API class A cement with 18 pct NaCl additive. The hole was completed 3¾ in. in diameter to a total depth of 1,414 ft, approximately 25 ft below the bottom of the 5.6-ft-thick Blue Creek Coalbed.

Treatment Implementation (February 25, 1981)

A suite of strata characterization geophysical logs were run prior to stimulation. A jetting tool was positioned approximately 15 ft above the bottom of the Blue Creek Coalbed and rotated to create a horizontal slot in the coalbed exposed in the wellbore. Very little coal was seen in the fluid returns, therefore the tool was raised 3 ft after 15 min of jetting. A good coal return was observed at the new position. Water injection rates were 2.5 bbl/min with sand added at a rate of 0.8 lb/gal, for a total fluid volume of 145 bbl and 2,000 lb of sand. Surface pressures during jetting averaged 1,500 lbf/in²(ga).

The stimulation treatment for borehole DHM-5 consisted of 75-pct-quality nitrogen foam without sand proppant and was conducted through tubing with a packer set in the casing. Formation break was reported by Boyer (17) at 800 lbf/in²(ga) surface and 1,400 lbf/in²(ga) bottom hole pressures. A continuous stimulation treatment record for this hole is not available. A 50,600-gal treatment was designed for this borehole, but only 40,866 gal of foam was used because of the excessive amount of

water used in the jet slotting procedure and a problem with supplying water to the hilltop location. Pumping rates were 7 bbl/min throughout the treatment, with surface pressures ranging from 800 to 950 lb/in²(ga). Fluorescent paint was also added to the stimulation fluids at a rate of 1.5 gal per 5,000 gal to aid in underground delineation of the stimulation fluid movements. At the time of stimulation, borehole DHM-5 was approximately 2,900 ft ahead of mining.

Gas production from the no-proppant treatment in this borehole averaged 22,000 ft³/d, which was more than the 2,000- to 3,800-ft³/d rates Boyer (17) stated was typical of unstimulated wells in the area, but was about half of the average 50,000-ft³/d rate for 17 stimulated wells at the mine site where sand proppant was used. Additional detailed production information can be obtained from reference 17.

Underground Observations

Mining approached to within 80 ft of the projected location of the borehole (fig. 15) but did not intercept the borehole. The exposure of the fracture on the face nearest the borehole was described by Boyer (17) as a vertical zone 4 to 5 in. in width in the upper third of the coalbed. The fracture was only identifiable by the presence of the fluorescent paint coating a series of parallel cleat with no apparent openings. This fracture was again observed entering the southern edge of the southeast corner of the pillar due east of the face location (fig. 15). At this point the fracture consisted of only a few paint-coated vertical cleat in a 1-in-wide zone confined to the upper 1 ft of the coalbed. This fracture continued through two adjacent pillars to the east for a total length of 340 ft. The orientation of the observed vertical paint-coated fracture

was nearly east-west. No evidence of a horizontal fracture component was reported.

One paint-coated fracture in the roof strata above the coalbed was identified by the ultraviolet light to extend from the coal fracture observed nearest the borehole. Water was observed flowing slowly from the roof rock and coal in the vicinity of the fracture at this face. According to Boyer (17), the paint-coated roof penetration appeared to be following a preexisting joint in the roof rock.

Tracing the roof penetration away from the face across the entry revealed two locations where the east-west trending roof joint intercepted and stairstepped along north-south trending roof joints. The roof penetration extended laterally in an arc to the southern edge of the coal pillar to the east and appeared to be an extension of the coal fracture previously described in the upper 1 ft of coal at this point. No additional exposure of this roof penetration was evident, including the site of the additional coal fracture exposure on the eastern margin of the same pillar. Apparently the roof penetration terminated somewhere in the corner of the pillar. The last observation of the roof penetration was approximately 150 ft from the borehole. According to Boyer (17), the stimulation did not appear to affect mining operations, and no roof, floor, or coal stability problems were observed in this area of the mine.

OAK GROVE MINE, BOREHOLE DHM-6 (17)

Borehole Completion

Borehole DHM-6 was drilled 6½ in. in diameter to a depth of 1,246 ft. The hole was to have been drilled to a point into the shale parting between the Mary Lee and Blue Creek Coalbeds so that the Blue Creek Coalbed could be completed open hole. However, the driller did not recognize drilling into the Mary Lee Coalbed and he continued drilling, eventually drilling through the Blue Creek Coalbed and approximately another 16 ft below before drilling was stopped. The hole was cased to the total depth with 4½-in.-OD casing, which was cemented in place with API class A cement with 18 pct NaCl additive. An additional 40 ft of 3½-in.-diameter hole was drilled below the casing before wireline geophysical logs were run to evaluate the hole and locate the target Blue Creek Coalbed. The Blue Creek Coalbed was identified at 1,238.6 to 1,244.4 ft for a total of 5.8 ft of coal, all inadvertently behind casing. The parting between the Mary Lee and Blue Creek Coalbeds was 10.8 ft and the Mary Lee Coalbed was only 1 ft thick at this location.

Treatment Implementation (December 20, 1980)

A jetting tool was positioned at 1,244 ft, near the bottom of the Blue Creek Coalbed interval to cut two vertical slots through the casing and cement to expose the coalbed to the wellbore. A good return of coal was obtained in the water circulated to surface during the jetting, indicating a successful penetration into the coalbed. Pressures recorded at the surface during the slotting varied between 1,800 to 2,000 lb/in²(ga). The water injection rate was 3 bbl/min with a sand rate of 2 lb/gal for a total fluid volume of 80 bbl of water containing 1,500 lb of sand.

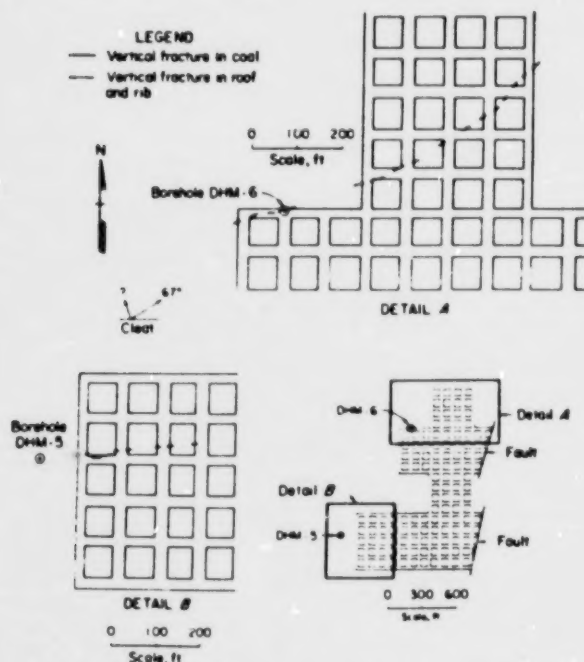


Figure 15.—Mine map, boreholes DHM-5 and DHM-6, with fracture orientations. (Modified from Boyer (17))

The stimulation treatment began with a 75-pct-quality nitrogen-generated foam pad at an injection rate of 3 bbl/min and was increased in 1-bbl/min increments until a rate of 7 bbl/min was reached (table 4). A surface breakdown pressure of 700 lb/in²(ga) was reported (17). A total of 40,320 gal of foam pad was injected into the coalbed before the addition of 20/40-mesh sand proppant. Sand was added to the treatment at a rate of 1 lb/gal of foam while the foam rate stayed constant at 7 bbl/min. A total of 10,000 lb of sand was injected with 10,248 gal of foam during the final 35 min of pumping. Fluorescent paint was included with the treatment fluids. Surface pressures ranged from 800 to 900 lb/in²(ga) during the treatment.

Bottom hole pressures were continuously monitored during the stimulation treatment and are presented in figure 16. The bottom hole pressure increased steadily as the injection rate was increased, up to a rate of 4 bbl/min. At the 4-bbl/min rate the bottom hole pressure began to decline, even though the injection rate was progressively increased to 7 bbl/min. The bottom hole pressure increased only slightly when sand was first added to the treatment, but as sand injection continued, the pressure gradually dropped. The bottom hole pressure never returned to the maximum observed at the 4-bbl/min injection rate.

Table 4.—Summary of treatment chronology, borehole DHM-6 (17)

Time, min	Foam		Av bottom hole pressure, lb/in ² (ga)
	Injection rate, bbl/min	Volume, gal	
0 to 15	3	1,890	1,050
15 to 30	4	2,520	1,100
30 to 60	5	6,300	1,090
60 to 90	6	7,560	1,080
90 to 165	7	22,050	1,090
165 to 200	7	10,248	1,080

¹With sand added.

At the time of the stimulation, the borehole was located approximately 1,800 ft from active mining. Gas production from this borehole quickly reached 80,000 to 90,000 ft³/d and remained at this rate until March 16, 1981 (approximately 3 months after stimulation) when the east wing of the fracture was intercepted by mining at a location 160 ft northeast of the borehole (fig. 15). Production rates declined and were somewhat erratic after the mine-through of the east fracture wing, but generally ranged between 40,000 to 50,000 ft³/d until mining came very close and finally intercepted the borehole in early 1982. A more complete production history can be found in reference 17.

Underground Observations

Inspection of the well casing exposed by mining of the Blue Creek Coalbed revealed two vertical slots, 180" apart, cut in a northeast-southwest direction. A vertical fracture zone, 1 to 2 in. in width and confined to the upper 3 ft of the coalbed, was observed on the rib 2 ft to the northeast of the borehole. The fracture zone consisted of multiple vertical planes, a maximum of 1/4 in wide, propped with sand and coated with fluorescent paint. A single vertical sand-and-paint-coated fracture was also observed in the 2 ft of exposed roof rock at this location. Boyer (17) commented that it was "difficult to ascertain if the roof fracture was produced by the hydraulic stimulation or if the hydraulic stimulation opened a pre-existing roof joint." The northeast wing of the fracture was next observed on the rib around the corner from the previous exposure, and 180 ft from the borehole (fig. 15). The fracture was still confined to the upper 3 ft of the coalbed, but consisted of a 10- to 12-in-wide zone of multiple vertical paint-coated face cleat surfaces with no sand proppant. The fracture terminated at the characteristically smooth, mirrorlike slickensided surface at the top of the coalbed. No penetration of the roof was observed at this

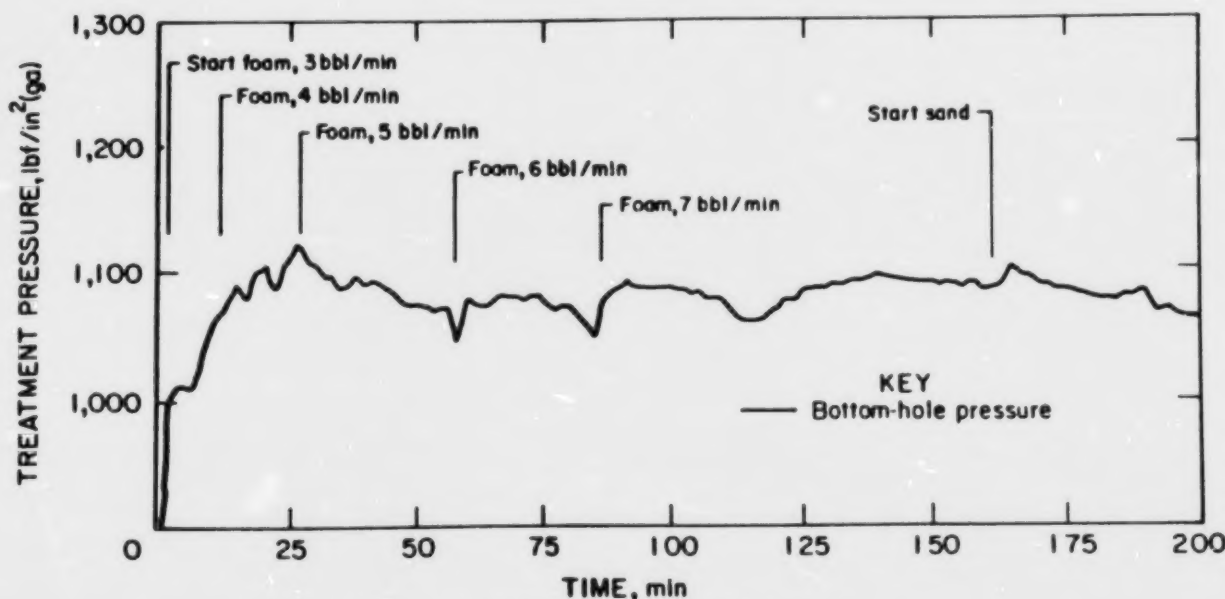


Figure 16.—Treatment record, borehole DHM-6 (17).

location or any further along this fracture wing. The fracture could be traced an additional 450 ft across the five entries to the northeast, finally entering the barrier pillar 630 ft from the borehole, the longest vertical fracture traced underground to date. The fracture was generally uniform in character across the five entries, consisting of multiple paint-coated vertical cleat surfaces in a narrow zone at the top of the coalbed. At the last observation point the fracture consisted of a 1- to 2-in-wide zone in the upper 1 ft of the coalbed. The northeast fracture wing varied slightly in orientation through its 630-ft traceable length, but generally followed an orientation of N 67° E, essentially the same orientation measured for face cleat at the mine site. No horizontal component was observed at any location along or in the vicinity of this fracture wing.

The vertical fracture wing to the southwest was observed on the rib 95 ft from the borehole (fig. 15). The

fracture consisted of multiple vertical planes propped with sand and contained fluorescent paint in a 1- to 2-in-wide zone confined generally in the upper 3 ft of the coalbed. Small amounts of sand and paint were also seen in the lower portion of the coalbed. The fractures were oriented similarly to those in the northeast wing and appeared to be face cleats. No horizontal component was observed at this location. A single sand-and-paint-coated fracture, $\frac{1}{4}$ in wide, was present in the mine roof at the wellbore and was traced to the mine face 95 ft away. Small amounts of gas and water were flowing from the roof fracture near the wellbore, and from the fracture at the coal face. Boyer (17) concluded after detailed observations of the mine roof that this roof fracture was probably a preexisting roof joint that had been penetrated by the stimulation fluid and proppant. He also stated that "no roof or coal stability problems were experienced in this area of the mine."

PITTSBURGH COALBED, PENNSYLVANIA

VESTA NO. 5 MINE, BOREHOLE USBM-4 (8)

Borehole Completion

Borehole USBM-4 was drilled 6½ in. in diameter to a depth of 597 ft. The only casing in the hole was 20 ft of 7½-in-diameter surface casing at the top of the hole. The hole was not cased to the coalbed because of the short time before anticipated mine-through of the borehole, which would preclude gas production; therefore, the expense of casing the hole did not seem justified. The Pittsburgh Coalbed was located at a depth of 588 to 594 ft for a total coal thickness of 6 ft. The total depth of the hole was 3 ft below the base of the Pittsburgh Coalbed. The immediate roof rock at the borehole was shale, and the floor rock was a calcareous shale. At the time of completion the borehole was 500 ft in advance of mining.

Treatment Implementation (April 17, 1974)

Borehole USBM-4 was treated through 2½-in-diameter tubing with an open hole packer set at 585 ft, 3 ft above the Pittsburgh Coalbed. The stimulation treatment record for this borehole is shown on figure 17. The

treatment design to be implemented was to use gelled water and sand to create two fracture wings, each 387 ft long, of which 219 ft would be propped by a sand pack, 5 ft high and 0.22 in wide. A 2,000-gal gelled water pad was first injected into the coalbed to control leakoff. Formation break was reported at a surface pressure of 500 lb/in²(ga).

A pumping rate of 10.5 bbl/min was maintained throughout the treatment. Following the pad, an additional 5,300 gal of gelled water and 3,500 lb of 10/20-mesh sand (3,000 lb coated with fluorescent dye followed by 500 lb of uncoated sand) was injected into the coalbed according to the schedule shown in table 5. It should be noted that sand injection rates reported by Elder (8) in table 5 when totaled (2,637 lb) do not match the 3,500 lb

Table 5.—Summary of treatment chronology, borehole USBM-4 (8)

Fluid volume, gal	Sand rate, lb/gal
2,000 (pad)	NAP
1,300	0.25
650	50
1,750	75
1,350	50
250 (flush)	NAP
NAP Not applicable	

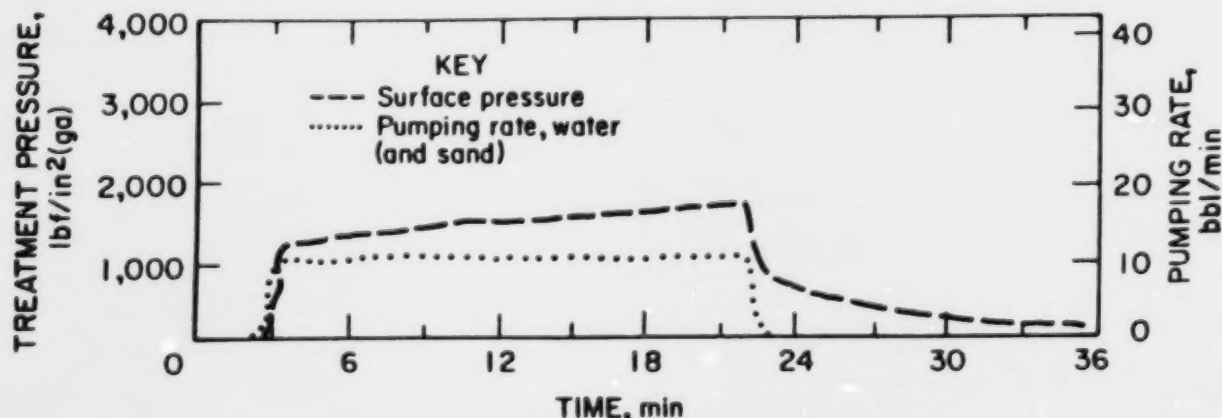


Figure 17.—Treatment record, borehole USBM-4. (Modified from Elder (8))

total sand weight reported in his text. The discrepancy may have resulted from inaccurate rates and or volumes for the various stages as determined at the blender. A total of 7,300 gal of 75 lb per 1,000 gal gelled fluid was used in the treatment. Surface treatment pressures averaged about 1,550 lbf/in²(ga) with a maximum surface pressure of nearly 1,800 lbf/in²(ga) recorded.

Underground Observation

Borehole USBM-4 was mined through on June 12, 1974, approximately 2 months after stimulation. The area of the borehole intercept is shown in map view on figure 18. Elder (8) reported that approximately 20 ft of sand-filled vertical fracture was followed head-on as mining progressed to the east and then intercepted the borehole (location 1, figure 18). The fracture extended from the roof to floor and ranged from $\frac{1}{8}$ to $\frac{1}{2}$ in. in width (fig. 19). The fracture appeared to have propagated along face cleats with an orientation of N 70° W. The fracture was essentially vertical, but several small offsets where the fracture was diverted laterally along bedding planes in the coal gave an apparent inclination to the fracture. Three vertical sand-filled fractures invading butt cleats on a N 20° E azimuth were observed on the north rib approximately 2 ft from the borehole (location 2, figure 18). These fractures deviated short distances laterally on bedding planes, producing a stairstepping pathway that resulted in an arcuate pattern in gross appearance (fig. 20). The primary vertical fracture in the butt cleat direction was a maximum of $2\frac{1}{2}$ in wide, and the two thinner fractures averaged approximately $\frac{1}{2}$ in wide. The primary butt cleat fracture extended from roof to floor and the two thinner fractures extended from the roof and terminated before reaching the floor.

There was no apparent butt cleat fracture observed on the rib south of the borehole nor was there any face cleat fracture reported to the east of the borehole. The maximum extent of the butt cleat fracture into the northern rib is not known, but the maximum extent of 20 ft reported for the face cleat fracture is far short of the designed 219 ft of sand-propped fracture length. There were no roof penetrations or adverse mining conditions

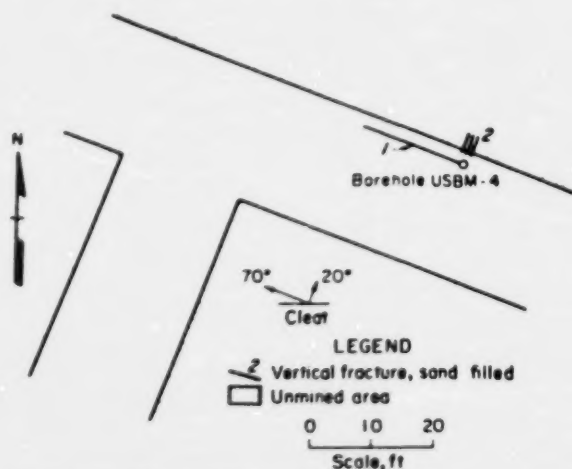


Figure 18.—Mine map, borehole USBM-4, with fracture orientations. (Modified from Elder (8))

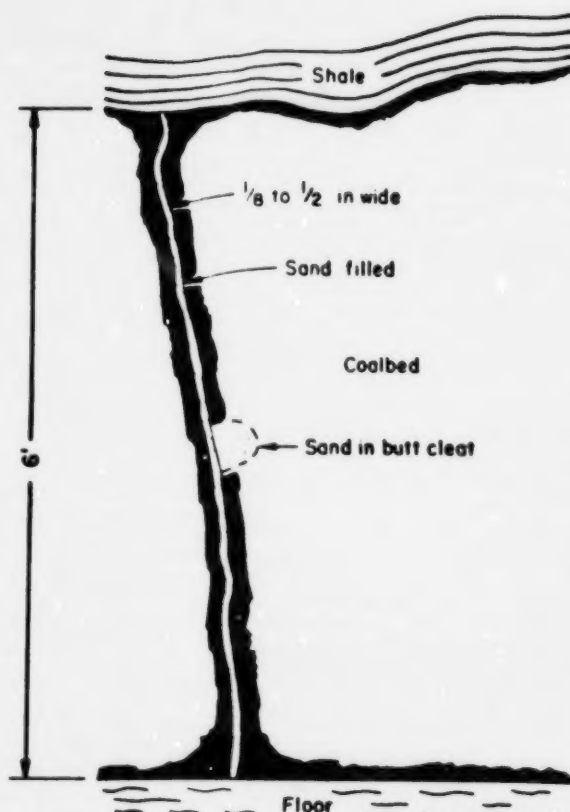


Figure 19.—Cross section of face cleat fracture in entry at location 1 near borehole USBM-4. (Modified from Elder (8))

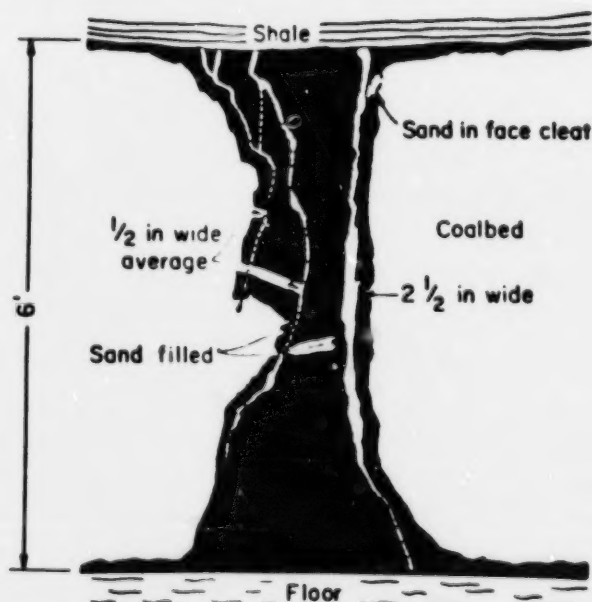


Figure 20.—Cross section of butt cleat fractures on rib at location 2 near borehole USBM-4. (Modified from Elder (8))

reported in the vicinity of borehole USBM-4. The site was revisited 2 yr after the initial mine-through, with no apparent roof or rib deterioration noted.

EMERALD MINE, BOREHOLE EM-5 (9-10)

Borehole Completion

Borehole EM-5 was drilled 6½ in. in diameter to a depth of approximately 810 ft (40 ft below the Pittsburgh Coalbed) and cased with 4½-in.-OD casing. The bottom of the casing was set in the shale above the target Pittsburgh Coalbed, and a formation packer shoe on the end of the casing was used to keep the cement from reaching the coalbed below. The coalbed was 6 ft thick and was located from 764 to 770 ft in the borehole. Prior to stimulation and production, the formation packer shoe was drilled out to provide communications with the coalbed below. At the time of completion, mining had not yet been initiated at the Emerald Mine. The borehole was located approximately 500 ft from the access shaft shown on figure 21.

Treatment Implementation (September 1, 1976)

Prestimulation strata characterization tests included gamma ray and neutron geophysical logs. A continuous stimulation treatment record is not available for this borehole. The treatment was initiated with a 900-gal water pad followed by 600 gal of 75-pct-quality nitrogen-generated foam. A formation breakdown pressure of 800 lb/in²(ga) was reported (10). A total of 31,500 gal of foam with 10,000 lb of sand (10/20- or 20/40-mesh) was injected into the coalbed. The average foam injection rate was 11.6 bbl/min, with the actual rate gradually increased as the sand rate was increased. The average surface treatment pressure was 1,375 lb/in²(ga). Gas production began relatively low but increased to a maximum of 117,000 ft³/d after 7 months of production. The rate declined to 60,000 to 75,000 ft³/d over the next several months. Details of the production history can be found in reference 10.

Underground Observations

A fracture wing from borehole EM-5 was intercepted by mining on July 21, 1978, approximately 2 yr after stimulation. The borehole location itself was not exposed by mining. The area of the fracture interception is shown in map view on figure 22. At the rib closest to the borehole, a single vertical sand-filled fracture up to 1½ in wide was observed extending from the floor to the roof (fig. 23). At the interface between the top of the coalbed and the overlying hard shale, the vertical fracture became horizontal. The sand-filled horizontal fracture was ¼ to ½ in wide, and at its maximum lateral extent was observed 50 ft from the presumed borehole location. The sand-filled vertical fracture could be traced across two entries of the mine, and at its maximum observed extent, it was approximately 140 ft from the presumed borehole location. The vertical fractures were propagated along face cleats, which averaged N 68° W in orientation. The width of the vertical sand-filled fracture varied from 1½ in at the exposure nearest the borehole to only a hairline at its maximum extent, where it was only partially filled with sand. No evidence of penetration of the roof strata was observed associated with the stimulation treatment at borehole EM-5.

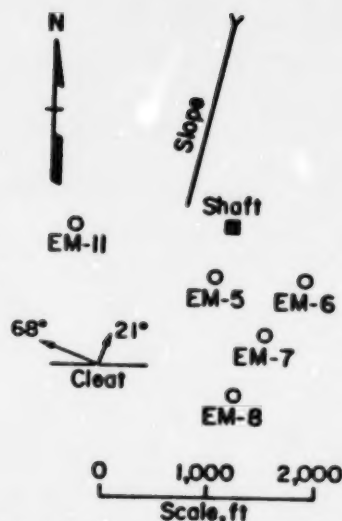


Figure 21.—Map of Emerald Mine area with locations of boreholes EM-5, EM-6, EM-7, EM-8, and EM-11. (Modified from Lambert (9))

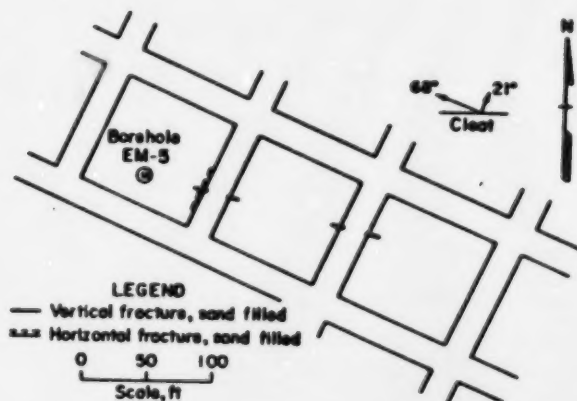


Figure 22.—Mine map, borehole EM-5, with fracture orientations. (Modified from Lambert (9))

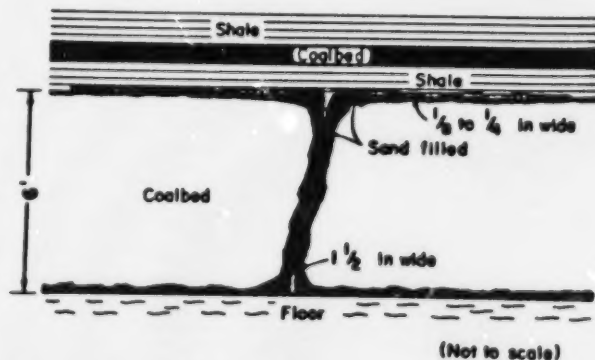


Figure 23.—Cross section of fractures on rib closest to borehole EM-5. (Modified from Lambert (9))

EMERALD MINE, BOREHOLE EM-6 (9-10)

Borehole Completion

Borehole EM-6 was drilled 6½ in. in diameter to a depth of approximately 628 ft (40 ft below the base of the Pittsburgh Coalbed) and cased with 4½-in.-OD casing. The bottom of casing was set in the shale above the target Pittsburgh Coalbed, and a formation packer shoe on the end of the casing was used to keep the cement from reaching the coalbed below. The coalbed was 6 ft thick and was located from 582 to 588 ft in the borehole. Prior to stimulation and production, the formation packer shoe was drilled out to provide communication with the coalbed below. At the time of completion, mining had not yet been initiated at the Emerald Mine. The borehole was located approximately 900 ft from the access shaft shown in figure 21.

Treatment Implementation (April 15, 1976)

Prestimulation strata characterization tests included gamma ray and neutron wireline geophysical logs. The continuous stimulation treatment record for this borehole is shown in figure 24. The treatment was initiated with a 900-gal water pad followed by 600 gal of 75-pct-quality nitrogen-generated foam. A formation break pressure of 950 lb/in²(ga) was reported. A total of 29,200 gal of foam with 14,000 lb of sand (10/20- or 20/40-mesh) was injected into the coalbed.

The average foam injection rate was 17.7 bbl/min, with the actual rate gradually increasing as the sand rate increased. The average surface treating pressure was 1,500 lb/in²(ga). It was reported by Steidl (10) that a screenout occurred near the planned end of the treatment causing only a slightly premature termination of the stimulation. Maximum surface pressure of about 2,150 lb/in²(ga) was reached at the start of sand injection and

again at the end of the treatment when the screenout was reported.

Gas production was generally erratic in the beginning, however, a maximum rate of 110,000 ft³/d was reported 1 yr after production began. The rate gradually declined to near 25,000 ft³/d at the termination of production prior to mine-through. Details of the production history can be found in references 9 and 10.

Underground Observations

Borehole EM-6 was mined through on November 9, 1978, approximately 2½ yr after stimulation. The area of the borehole interception is shown in map view in figure 25. A single vertical sand-filled fracture, approximately 2½ in wide and extending from the base of the coalbed to a point about 3 ft above the base was observed on the west rib, 10 ft away from the intercepted borehole (fig. 26). The vertical fracture was propagated along face cleats and several discontinuous sand-filled fractures extended to the top of the main coal bench. Several additional thin discontinuous sand-filled vertical fractures paralleled the main fracture and were also developed in face cleats. The average orientation was N 68° W. One of the short vertical fractures penetrated the thin roof shale above the main coal bench and terminated at the base of the rider coal.

An inclined fracture split off from the main vertical fracture and stairstepped its way up to the top of the coalbed where it became horizontal for a short distance before inclining through the thin (8-10 in) roof shale and finally becoming horizontal at the base of the rider coal. The sand-filled horizontal fracture on top of the main coal bench was about ½ in thick and the horizontal fracture at the base of the rider coal was ¼ to ½ in thick. No additional exposures were observed for the fractures seen on the ribs adjacent to the borehole. The extent of the fractures into the ribs is unknown. No adverse mining conditions were encountered due to the penetration of the

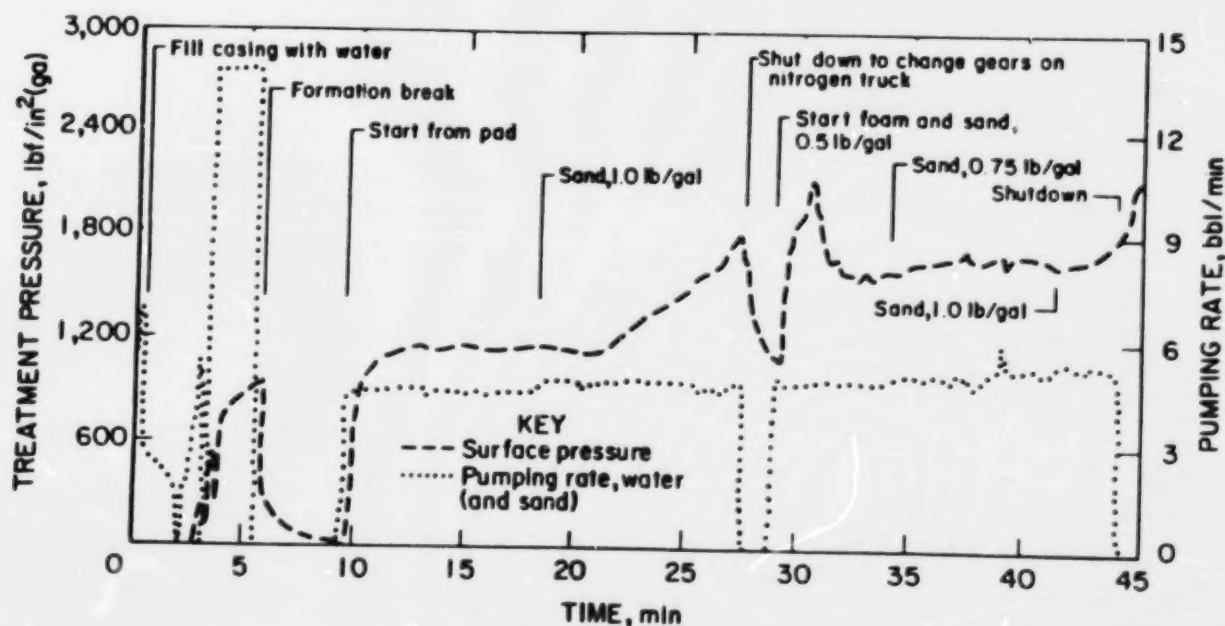


Figure 24.—Treatment record, borehole EM-6.

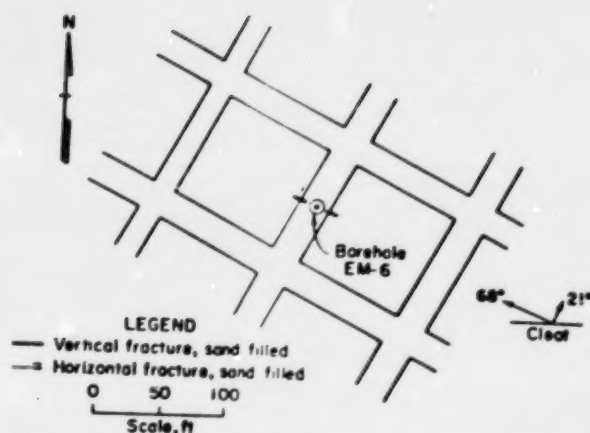


Figure 25.—Mine map, borehole EM-6, with fracture orientations. (Modified from Lambert (9))

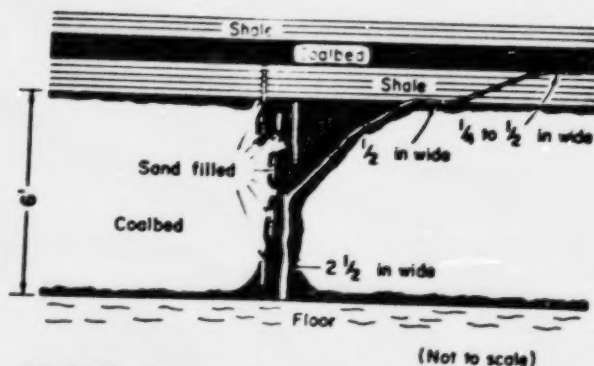


Figure 26.—Cross section of fractures on rib west of borehole EM-6. (Modified from Lambert (9))

strata overlying the main bench of the Pittsburgh Coalbed. In fact, common mining practice at the mine has been to extract the thin roof shale and part of the rider coal leaving a thin interval of the rider coal as the actual mine roof.

EMERALD MINE, BOREHOLE EM-7 (9-10)

Borehole Completion

Borehole EM-7 was drilled 6 1/4 in. in diameter to a depth of approximately 774 ft (40 ft below the base of the Pittsburgh Coalbed) and cased with 4 1/2-in.-OD casing. The bottom of casing was set in the shale above the target Pittsburgh Coalbed, and a formation packer shoe on the bottom of the casing was used to keep the cement from reaching the coalbed below. The coalbed was 6 ft thick and was located from 728 to 734 ft in the borehole. Prior to stimulation and production, the formation packer shoe was drilled out to provide communication with the coalbed below. At the time of completion, mining had not yet been initiated at the Emerald Mine. The borehole was located approximately 1,100 ft from the access shaft shown in figure 21.

Treatment Implementation (September 8, 1976)

Prestimulation strata characterization tests included gamma ray and neutron wireline geophysical logs. A continuous stimulation treatment record is not available for this borehole. The treatment was initiated with a 900-gal water pad followed by 600 gal of 75-pct-quality nitrogen-generated foam. A formation break pressure was not reported. A total of 29,000 gal of foam with 7,400 lb of sand (10/20- or 20/40-mesh) were injected into the coalbed. The average foam injection rate was 10.8 bbl/min, with the actual rate gradually increasing as the sand rate increased. The average surface treatment pressure was 1,400 lb/in² (ga). It was reported by Steidl (10) that a screenout occurred near the planned end of the treatment causing only a slightly premature termination of the stimulation. Gas production from the borehole began very high (120,000 ft³/d) reaching a maximum of 140,000 ft³/d after only 1 month of production. The rate eventually declined to about 6,000 ft³/d at the termination of gas production prior to mine-through. Details of the production history can be found in references 9 and 10.

Underground Observations

Borehole EM-7 was mined through on December 1, 1978, approximately 2 1/4 yr after stimulation. Vertical sand-filled fractures, 1/4 to 1/2 in wide, were reported a maximum of 18 ft laterally from the borehole. No horizontal fractures were present nor was there any penetration of strata above the main bench of the Pittsburgh Coalbed.

EMERALD MINE, BOREHOLE EM-8

Borehole Completion (10)

Borehole EM-8 was drilled 6 1/4 in. in diameter to a depth of approximately 692 ft (40 ft below the base of the Pittsburgh Coalbed) and cased with 4 1/2-in.-OD casing. The bottom of the casing was set in the shale above the target Pittsburgh Coalbed, and a formation packer shoe on the end of the casing was used to keep the cement from reaching the coalbed below. The coalbed was 6 ft thick and was located from 646 to 652 ft in the borehole. Prior to stimulation and production, the formation packer shoe was drilled out to provide communication with the coalbed below. At the time of completion, mining had not yet been initiated at the Emerald Mine. The borehole was located approximately 1,600 ft from the access shaft shown on figure 21.

Treatment Implementation (September 15, 1976) (10)

Prestimulation strata characterization tests included gamma ray and neutron wireline geophysical logs. The continuous stimulation treatment record for this borehole is shown in figure 27. The treatment was initiated with a 900-gal water pad followed by 600 gal of 75-pct-quality nitrogen-generated foam. A formation break was not reported. A total of 42,000 gal of foam with 12,800 lb of sand (10/20- or 20/40-mesh) were injected into the coalbed. The average foam injection rate was 10.8 bbl/min, with

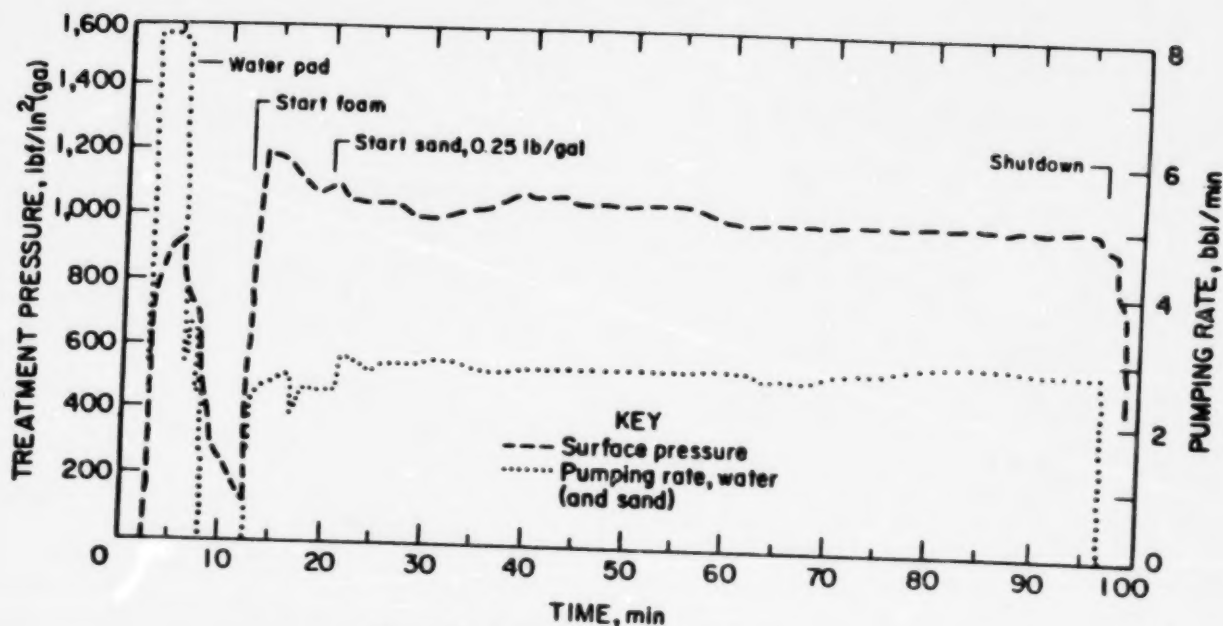


Figure 27.—Treatment record, borehole EM-8.

the actual rate gradually increasing as the sand rate increased. The average surface treatment pressure was 1,050 lb/in²(ga). Gas production from borehole EM-8 was never very high, with the maximum rate of 21,000 ft³/d being achieved approximately 1 month after stimulation. The rate eventually declined to only 2,000 ft³/d prior to the termination of gas production just before mine-through. Details of the production history can be found in reference 10.

Underground Observations

The area around borehole EM-8 was mined through in the spring of 1979, approximately 2½ yr after stimulation. A detailed underground investigation was not conducted until the summer of 1985 when the authors of this report visited the site. Vertical sand-filled fractures were observed at four locations on the eastern rib of the pillar presumed to contain the borehole (fig. 28). At location 1, the sand-filled fracture extended approximately 3.8 ft into the main bench of the coalbed from the top (fig. 29). At the top of the coalbed the fracture was ¼ in wide and tapered to only the width of a single sand grain at the bottom. The fracture was propagated along a face cleat with an orientation of N 65° W. The fracture penetrated up into the shale above the main bench where it intercepted and propagated along an inclined fracture which was about ¼ in wide. A second inclined fracture close to and nearly parallel to the first was also filled with sand. Both of these fractures terminated at the top of the shale, at the interface with the overlying rider coal. The two inclined fractures in the thin (0.9 ft) shale were very similar in appearance to other non-sand-filled naturally occurring fractures in the area, and are assumed to have existed prior to and been invaded by the stimulation treatment. A second short vertical sand-filled fracture, one sand grain thick, extended down from the top of the coalbed approximately 1 ft into the coalbed at location 1.

No adverse mining conditions were reported nor were deteriorated roof conditions observed at this location, even after exposure of the fracture for 6 yr. As noted earlier, the common mining height at this mine includes the shale unit that was penetrated by the stimulation treatment.

The vertical sand-filled fracture at location 2 (fig. 30) was contained in the lower 2 ft of the coalbed and turned horizontal above a distinctive bony coal band. The vertical fracture also stairstepped short distances along bedding planes in the coalbed before turning completely horizontal. The vertical fracture was ¼ in wide and was the same thickness but discontinuous after turning horizontal. The vertical fracture was contained in a face cleat with an orientation of N 72° W. There also seemed to be a sand-filled butt cleat fracture at this location, but the exposure was such that it could not be conclusively delineated.

At location 3, two short (1½-ft) en echelon sand-filled vertical fractures extended from the roof approximately 3 ft down into the coalbed (fig. 31). The upper fracture was ¼ in wide near the top and widened to ½ in at the bottom. The second fracture was ¼ in wide at the top and narrowed to ¼ in at the bottom. Both fractures were propagated along face cleats with an orientation of N 66° W. No roof penetration or horizontal fractures were observed at this location.

At location 4, a sand-filled fracture was essentially vertical but did stairstep short distances on horizontal bedding planes in the coal (fig. 32). Above the distinct shale bands the fracture was about ¼ in wide, and narrowed slightly to ¼ in below the shale bands. The fracture extended from the floor to the top of the coalbed, and penetrated 0.15 ft into the overlying shale. No adverse mining conditions were experienced as a result of this minor penetration of the thin shale bed. The fracture at this location was predominately propagated along face cleats with an orientation of N 67° W. Short sand-filled butt cleat fractures were also observed. This location, at 55

ft, was the maximum observed extent of a vertical fracture from the presumed location of the borehole. No horizontal fracture was present at this location.

An extensive investigation of the pillars surrounding the one presumed to contain the borehole did not reveal any additional vertical fractures. Even though only minor exposures of butt cleat fractures were observed, a substantial portion of the fracture must have stairstepped along the butt cleat because it is otherwise impossible to extend the four nearly parallel face cleat oriented fractures back to the presumed location of the borehole.

Horizontal fractures were observed at several locations, generally to the south of the borehole (fig. 28). The fractures were found on horizontal bedding planes in the thin shale below the rider (above the main bench), and on the shale-coal interface above and below the rider coal. At some locations, horizontal sand-filled fractures could be found on more than one of the described horizontal surfaces. The fractures ranged from $\frac{1}{16}$ in to essentially only a single sand grain in thickness. The maximum distance a horizontal sand-filled fracture was observed from the presumed location of the borehole was 115 ft.

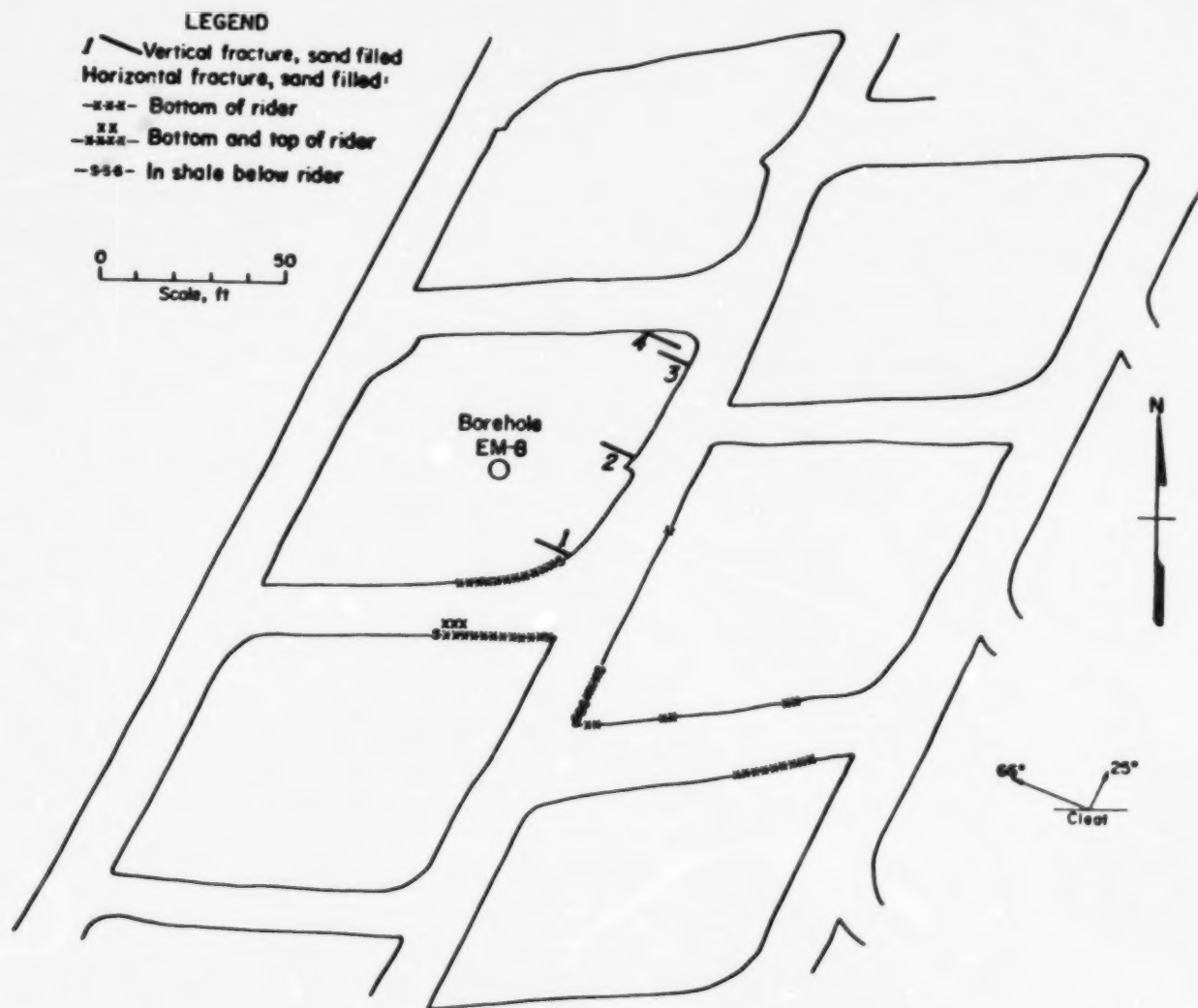


Figure 28.—Mine map, borehole EM-8, with fracture orientations.

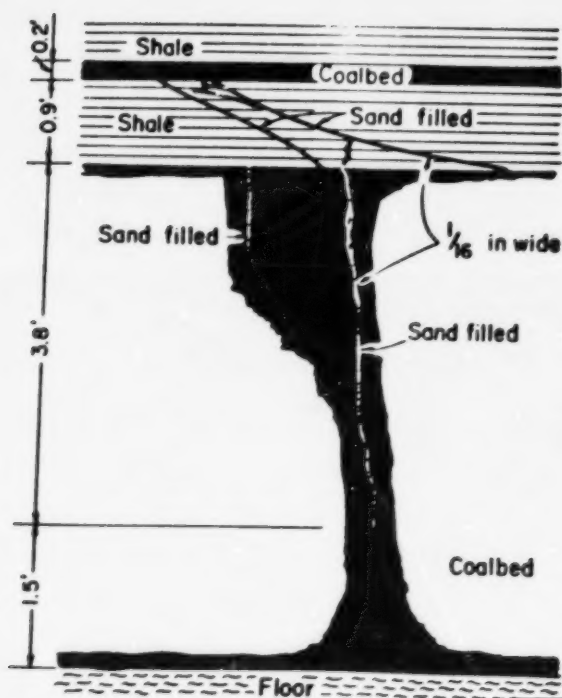


Figure 29.—Cross section of fractures on rib at location 1 near borehole EM-8.

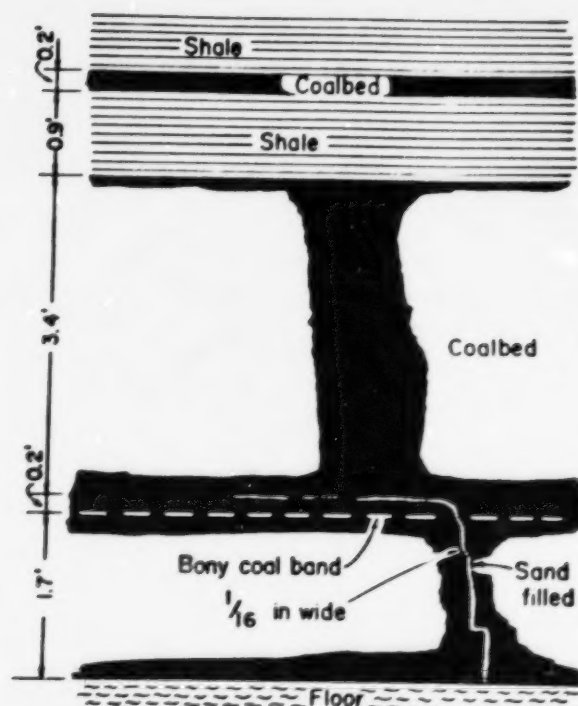


Figure 30.—Cross section of fractures on rib at location 2 near borehole EM-8.

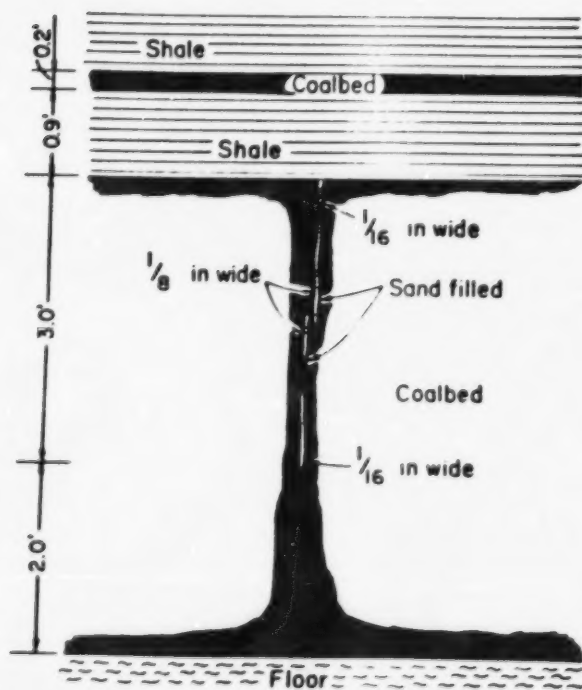


Figure 31.—Cross section of fractures on rib at location 3 near borehole EM-8.

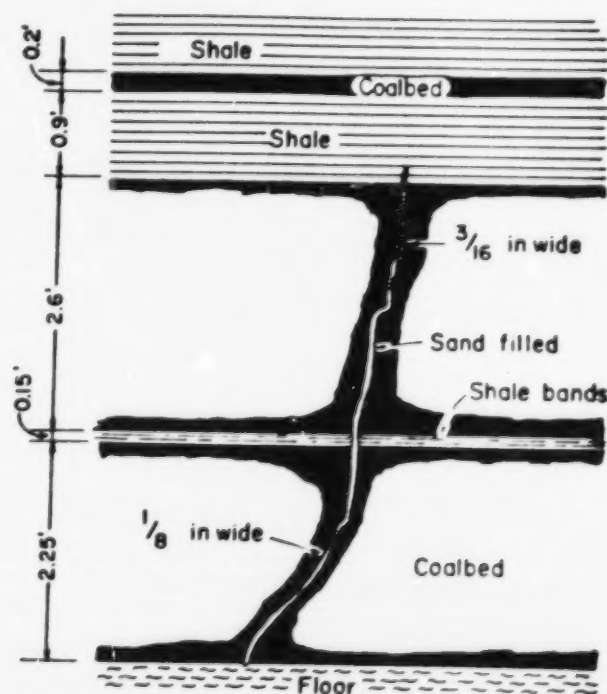


Figure 32.—Cross section of fracture on rib at location 4 near borehole EM-8.

EMERALD MINE, BOREHOLE EM-11 (9)

Borehole Completion

Borehole EM-11 was drilled to a depth of 725 ft, 6 ft below the base of the 6-ft-thick Pittsburgh Coalbed. The top of the coalbed was at a depth of 713 ft. The hole was cased and cemented to the top of the coalbed. At the time of completion, little mining had been completed at the Emerald Mine. The borehole was located approximately 1,000 ft from the bottom of the slope shown in figure 21.

Treatment Implementation (May 14, 1977)

Borehole EM-11 was stimulated with a patented treatment called a Kiel Frac (23). This treatment technique utilized multiple pressurization-depressurization cycles that would theoretically create long vertical fractures along transverse directions, then prop the fractures with reservoir rock material that sloughs off during the treatment. The treatment fluid was water with 80/100- and 20/40-mesh sand. The sand was added primarily to control fluid loss. In addition, the patent (23) claims that "the fracture formed in the producing formation may be confined to the upper portion of the producing formation where lithology (overlying rock) will limit upward growth of the fracture and sand properly scheduled will limit the downward growth of the fracture."

The treatment on borehole EM-11 used over 54,600 gal of water, injected at a rate of about 19 bbl/min, and 10,000 lb of sand proppant. Surface pressures during the pressurization cycles averaged about 1,200 lb/in² (ga). A continuous treatment record is not available for this borehole. Gas production from this borehole began relatively high (over 40,000 ft³/d) and reached a maximum of almost 87,000 ft³/d within a month after production began. The production rate gradually declined to 3,000 ft³/d prior to termination of production before mine-through.

Underground Observations

Sand-filled fractures from borehole EM-11 were intercepted in June of 1978, approximately 1 yr after stimulation. The borehole was not mined through. The area of the fracture interception is shown in map view in figure 33. The only observed fractures were horizontal and were located in the rider coal above the main bench of the Pittsburgh Coalbed (fig. 34). One of the fractures curved up into the shale overlying the rider coal.

The penetration of the upper shale bed at this location may have been influenced by the presence of a clay vein. Clay veins are commonly characterized by planes of weakness at their interface with surrounding strata. The location of the exposures of horizontal fractures was approximately 110 ft from the presumed location of the borehole. No other exposure of sand-filled fractures was observed, and no adverse mining conditions were reported related to the presence of the horizontal fractures in the rider coal, which is commonly mined along with the main bench. Since borehole EM-11 was not completely mined around, additional sand-filled fractures may have been created, but not mined through. Vertical fractures that would have extended from the borehole in the face cleat direction could have been intercepted by the mine

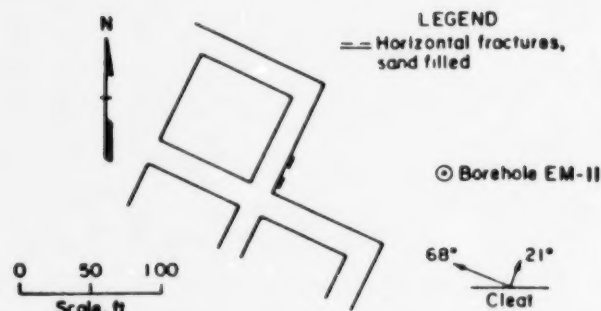


Figure 33.—Mine map, borehole EM-11, with fracture orientations. (Modified from Lambert (9))

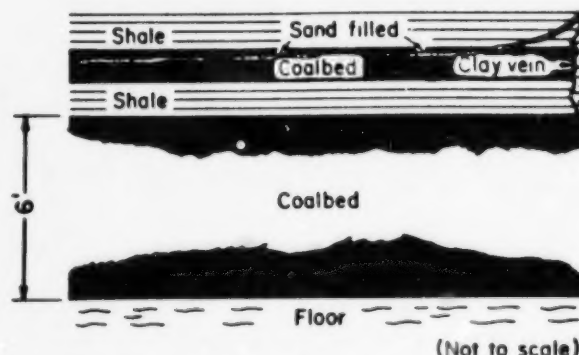


Figure 34.—Cross section of fractures on rib near borehole EM-11. (Modified from Lambert (9))

development available for inspection, unless significant stairstepping (as the Kiel Frac was specifically supposed to accomplish) along the butt cleats had occurred.

CUMBERLAND MINE, BOREHOLE CNG-1034

Borehole Completion (24)

Borehole CNG-1034 was drilled 7-in. in diameter to a depth of 825 ft, a point 63 ft below the base of the Pittsburgh Coalbed. The target Pittsburgh Coalbed was 8 ft thick and located from 754 to 762 ft in the borehole. The hole was cased with 5 1/2-in.-OD casing to a depth of 755 ft, 1 ft into the top of the coalbed. A retrievable packer was set to prevent cement infiltration into the coalbed below casing. The distance from the borehole to active mining is not known.

Treatment Implementation (May 5, 1977) (24)

Prestimulation strata characterization tests included natural gamma ray, density, neutron, temperature, and caliper wireline geophysical logs along with a driller's log and drilling time log. The hole was stimulated with 75-pct-quality nitrogen-generated foam with sand proppant. Prior to the stimulation treatment a mixture of Cal-seal¹ and 3-in. limestone chips was used to fill the

¹Reference to specific products does not imply endorsement by the Bureau of Mines.

bottom of the hole below the Pittsburgh Coalbed to keep the treating fluids from invading the strata below. The treatment was initiated by pumping a 2,000-gal foam pad at a rate of 10 bbl/min. A formation break at 800 lb/in²(ga) surface was reported for this treatment. A continuous stimulation treatment record is not available for this borehole, but a summary of the general treatment chronology is presented in table 6.

Table 6.—Summary of treatment chronology, borehole CNG-1034 (24)

Foam vol., ¹ gal	Sand		
	Rate lb/gal	Size	Weight, lb
2,000 (pad)	NAP	NAP	NAP
2,000	0.5	Minus 100 mesh	1,000
2,000	1.0	do	2,000
4,000	1.0	20-40 mesh	4,000
10,000	1.5	do	15,000
1,000	1.5	do	1,500
640 (flush)	NAP	NAP	NAP

NAP Not applicable

¹Constant injector, rate of 10 bbl/min

²Dye coated

Foam was injected at a constant rate of 10 bbl/min throughout the treatment, with sand added at an initial rate of 0.5 lb/gal and gradually increased in 0.5-lb/gal increments to a maximum rate of 1.5 lb/gal. Minus 100-mesh sand was added at the beginning of the treatment as a fluid loss agent before the 20-40-mesh sand was added as a proppant. Near the end of the treatment, 1,500 lb of dye-coated sand was added to the stimulation fluid. A total of 21,840 gal of foam, 640 gal of water (flush at end), and 23,500 lb of sand were used in the stimulation treatment. The average surface treatment pressure was reported to be 1,200 lb/in²(ga). The computer-designed treatment was to propagate a vertical fracture with a length of 2,764 ft. Gas production from this hole was low, with the maximum rate of 5,400 ft³/d reported approximately 1 month after stimulation.

Underground Observations (19)

Fractures from borehole CNG-1034 were intercepted in 1981, approximately 4 yr after stimulation. A vertical fracture filled with sand was observed 85 ft to the east of the presumed location of the borehole (fig. 35). The fracture extended from the floor to the roof along face cleat



Figure 35.—Mine map, borehole CNG-1034, with fracture orientations (19).

and varied in width from $\frac{1}{8}$ to $\frac{1}{4}$ in. The fracture could be traced in the roof coal across the entry in the face cleat direction to where it was exposed as a vertical fracture on the corner of the opposite rib. The fracture continued through the corner of the rib, exiting on the opposite side. The fracture could be traced an additional 10 ft across the entry in the roof coal, approximately 130 ft from the presumed location of the borehole. No penetration of the strata above the coalbed was reported along the exposed length of the fracture.

The entry adjacent to the pillar that contained the borehole would also be expected to contain an exposure of the fracture since it is closer to the borehole than the exposures previously described, however, a roof fall in this entry prevented direct observation of the area. Investigation of the roof fall revealed that it appeared to be related to a "roll" in the roof, which was characterized by highly disturbed bedding and abundant slickensided surfaces that characteristically have a lack of cohesiveness. Roof falls associated with similar geologic conditions are common at the mine site, therefore, the mine company personnel did not attribute the roof fall to the stimulation treatment. No evidence of a vertical fracture wing was seen to the west-northwest of the borehole, nor were any horizontal fractures reported at any location.

UPPER FREEPORT COALBED, PENNSYLVANIA

LUCERNE NO. 6 MINE, BOREHOLE RP-1 (12)

Borehole Completion

Borehole RP-1 was drilled with air, 7 in. in diameter to a depth of 675 ft, approximately 42 ft below the base of the target Upper Freeport Coalbed. The coalbed was 6.75 ft thick with a 5-in-thick shale parting near the top of the coal interval, and was located from 626.5 to 633.25 ft in the borehole. Prior to casing the borehole, sand was used to fill the hole to the top of the coalbed as an extra

precaution against cement penetrating the coal interval. The borehole was cased with 4½-in.-OD casing with a cement basket shoe on the bottom as the primary means of keeping cement from reaching the coalbed below the casing. The bottom of casing was set at 622 ft, 4.5 ft above the top of the coalbed. The bottom of casing was to be set closer to the top of the coalbed, but debris from running the casing fell on top of the sand pack preventing the casing from reaching the desired depth. The basket shoe and sand pack were drilled out to a depth of about 2 ft below the base of the coalbed in preparation for stimulation.

Treatment Implementation (January 21, 1982)

Prestimulation strata characterization tests on this borehole were quite extensive. Gamma ray and density wireline geophysical logs were obtained prior to casing the borehole. A series of in situ state of stress (ISSOS) tests were made in the borehole prior to setting casing. The ISSOS test is essentially a "mini-stimulation" treatment that introduces a small amount of fluid into isolated zones in the wellbore to determine a "breakdown" pressure and instantaneous shut-in pressure.

An ISSOS test in the roof rock above the coalbed (618 to 624 ft) produced two formation breaks, one at slightly less than 800 lb/in²(ga) bottom hole and a second at about 1,600 lb/in²(ga) bottom hole. The first break was interpreted to be a vertical fracture and the second a horizontal fracture (25). An instantaneous shut-in pressure (ISIP) of 750 lb/in²(ga) bottom hole was recorded. No definitive formation break was recorded for the coalbed interval tested (626 to 634 ft); the ISIP was only 490 lb/in²(ga) bottom hole, which was also the maximum pressure recorded during the test.

A series of mechanical property tests including triaxial compressive strength, Young's modulus, and residual friction angles and coefficients were conducted on coal and surrounding rocks from the mine site (25). In

general, the rock was found to have a higher strength than the coal. A stimulation treatment would be expected to stay in the coalbed if higher stress and/or higher strength rocks surrounded it, and the treatment pressures are only allowed to barely exceed the pressure necessary to "break" the coalbed. In this case, a high enough differential between the ISSOS test values and the laboratory derived strength data would appear to be present to keep the stimulation treatment in the coalbed if the treatment pressure was kept near the 490-lb/in²(ga) bottom hole ISIP value of the coalbed.

The treatment for borehole RP-1 was to be very small, using only 2,775 gal of 75-pct-quality nitrogen-generated foam (injected at a rate of 5 bbl/min) and 1,360 lb of 20/50-mesh sand proppant. The treatment was small because the hole was located only 250 ft from mining. The computer-designed treatment would have theoretically produced two sand-filled fracture wings each extending for a distance of 209 ft from the borehole. The coalbed was not jet slotted prior to the stimulation since it was presumed to have already been fractured during the ISSOS test.

During the initial stage of the treatment, while using clear water to initiate a fracture in the coal, treatment pressures steadily climbed and remained high. The continuous stimulation treatment record is shown on figure 36. Several cycles of pressure buildup and release

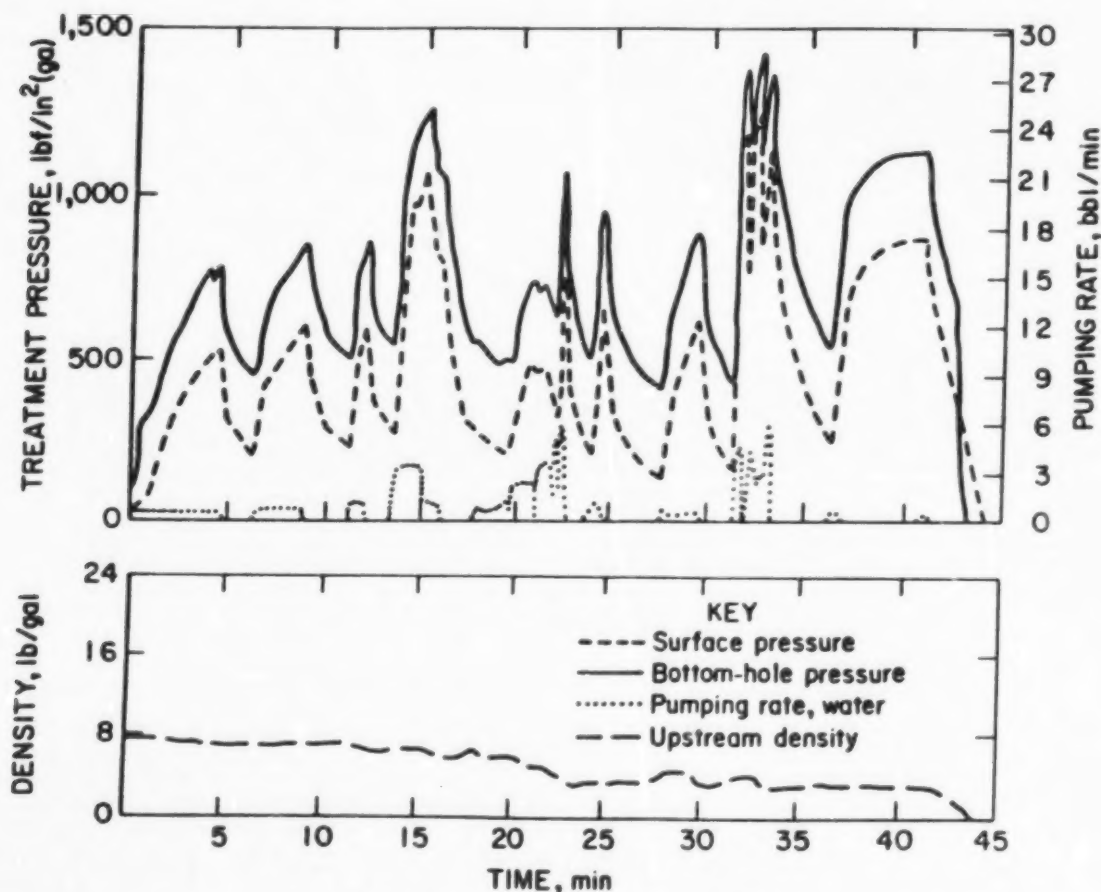


Figure 36.—Treatment record, borehole RP-1.

were made in an attempt to initiate a fracture, without success. A total of 1,600 gal of clear water containing green fluorescent paint was injected into the coalbed at rates ranging from 0.4 to 7.4 bbl/min. No sand proppant was included in the injected fluid. Maximum treatment pressures of 1,450 lb/in²(ga) bottom hole and 1,260 lb/in²(ga) surface were reached. Table 7 summarizes parameters for the stimulation.

Table 7.—Summary of treatment chronology, boreholes RP-1 through RP-3 (12)

Borehole and pigment color	Treatment	
	Stage	Vol. gal
RP-1 Green	Water pad	1,800
RP-2:		
Blue	Foam pad	10,000
Orange	Foam and 0.5 lb/gal sand	4,400
Yellow	Foam and 0.5 to 1.0 lb/gal sand	5,400
RP-3:		
Red	Foam pad	20,000
Blue	Foam and 0.5 lb/gal sand	2,500
Orange	Foam and 1.0 lb/gal sand	1,800
Yellow	Foam and 1.5 lb/gal sand	8,100

The borehole (plus boreholes RP-2, RP-3, KE-2, and SC-1 to be discussed later) was not completed for production since the project goals (BuMines contract J0333908) were only to stimulate and evaluate the effect of the stimulation treatment on the coalbed and surrounding strata.

Underground Observations

Borehole RP-1 was mined through on February 12, 1982, less than 1 month after stimulation. The borehole was found approximately 25 ft from the expected location determined by a deviation survey. Extensive mapping of existing geologic conditions was conducted near this borehole and the two companion boreholes (RP-2 and RP-3) prior to the stimulation and mine-through, to aid in interpreting the effect of the treatment on the coalbed and surrounding strata. A complete geologic interpretation of the area surrounding the three boreholes completed at this mine site can be found in the contract final report (12). Many visits were also made to the area expected to be influenced by the treatments as mining progressed towards the boreholes, so that any exposures on the advancing faces that would be lost by the continual advance would be documented. These multiple site visits, the use of fluorescent paint in the treatments, and the extensive mining around the three boreholes at the Lucerne No. 6 Mine make the underground characterizations (along with those of borehole TW-5 at the Oak Grove Mine) the most complete of the stimulation treatments observed.

Vertical fractures (25)^a from borehole RP-1 were found in the southeast rib adjacent to the intercepted borehole (fig. 37). Nine vertical fractures in a zone 8½ in

^aBoldface numbers in parentheses refer to labeled fracture traces in figure 37.

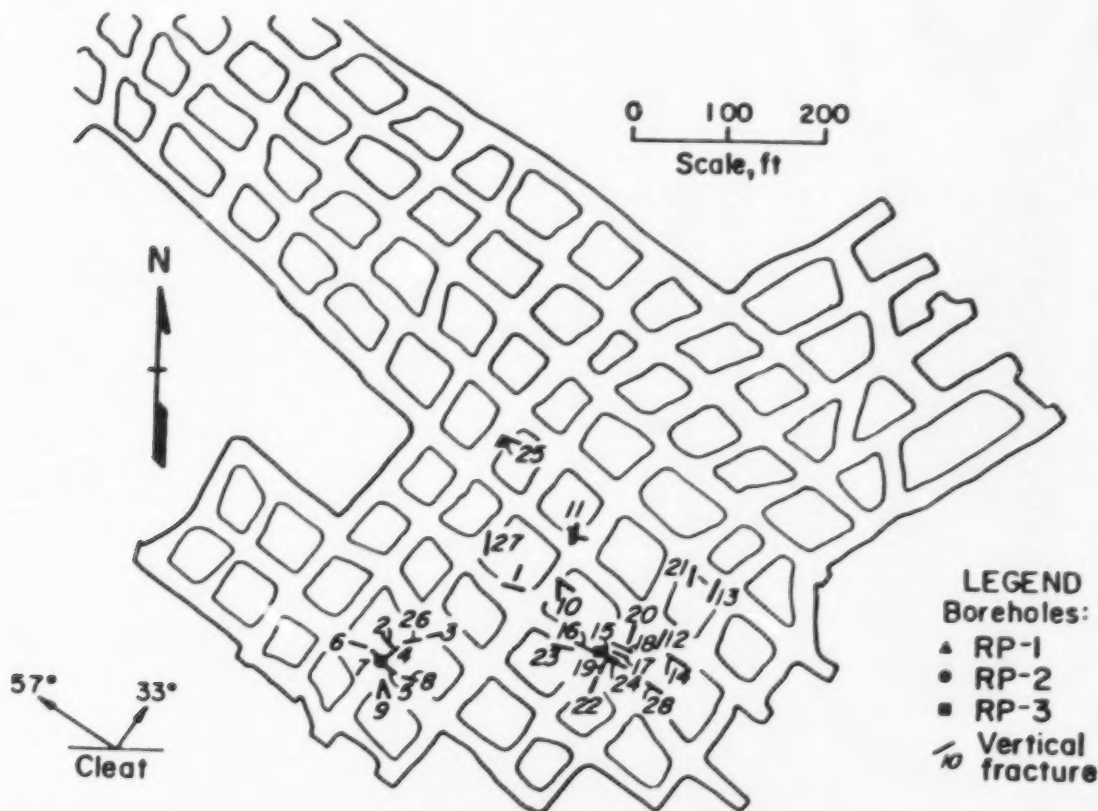


Figure 37.—Mine map, boreholes RP-1 through RP-3, with fracture orientations (12).

wide were present in the lower bench of coal below the shale parting (fig. 38). Two fractures contained sand that is assumed to have come from the sand placed at the bottom of the hole to protect the coalbed from cement invasion during the casing and cementing operation, since no sand was injected as part of the treatment.

Several of the vertical fractures extended through the shale parting and into the upper bench. Pink-paint-tagged grout that had been used to plug the hole for abandonment was found in the upper portion of fractures whose lower portion contained sand. All of the fractures were aligned parallel to the face cleat (N 76° W) and were coated with green paint. Fractures that contained only paint were visible using the ultraviolet light. They appeared to be no wider than the face cleat itself. The sand-filled fractures were approximately $\frac{1}{16}$ in wide. The most prominent vertical fracture could be traced with the ultraviolet light laterally across a small patch of roof coal remaining around the borehole. The fracture in the roof coal extended laterally from the borehole approximately 6 in to the nearby rib and 19 in into the entry in the opposite direction. The fluids followed a stairstep path alternating along the face and butt cleat in the roof coal, and did not travel along a single face cleat for more than 1 or 2 in. A summary of the orientation and fracture observations is given in table 8.

No evidence of horizontal fracturing was found close to borehole RP-1, but traces of the green paint were found on the roof shale at several points in the entry. The inferred horizontal extent of the paint pigment is shown in figure 39.

The vertical fractures either terminated at the roof or before reaching the roof. No evidence of roof penetration by the stimulation was observed. A small roof fall did occur at an intersection approximately 25 ft southwest of the borehole several days after being mined. The roof fall was attributed to the presence of compactional slips in the black shale roof. Examinations of the area with the ultraviolet light failed to detect any paint on any surface associated with the fall or around the fall. Similar roof falls related to slips are common in this area of the mine.

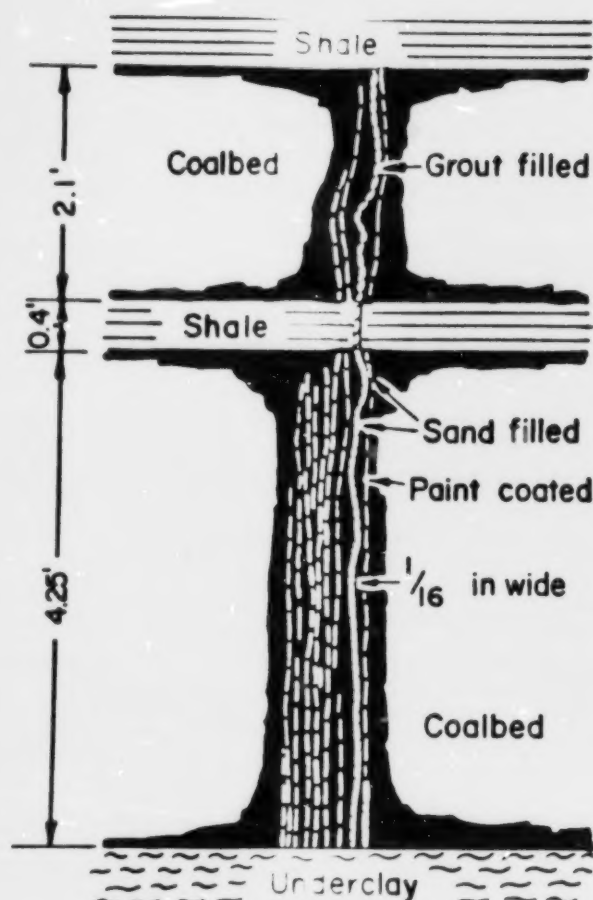


Figure 38.—Cross section of fractures at location 25 on the southeast rib near borehole RP-1. Modified from Murrie (12)

Table 8.—Orientation and description of vertical fractures, boreholes RP-1 through RP-3 (12)

Location ¹	Number of fractures	Quadrant bearing	Fracture filling materials ²	Location ¹	Number of fractures	Quadrant bearing	Fracture filling materials ²
1	1	N 70 E	R V	14	1	N 70 E	R V
2	1	N 46 W	O V	15	1	N 70 E	R V
3	1	N 69 E	O V	16	1	N 70 E	R V
4	1	N 59 E	O V S	17	1	N 70 E	R V
5	1	N 54 W	O V S C	18	1	N 70 E	R V
6	4	N 69 W	O V S C	19	3	N 70 E	R V
7	1	N 54 W	O V S C	20	1	N 70 E	R V
8	1	N 73 W	O V	21	9	N 70 E	R V
9	1	N 33 E	O V	22	1	N 70 E	R V
10	1	N 34 W	O V	23	1	N 70 E	R V
11	1	N 26 E	O V	24	1	N 70 E	R V
	1	N 9 E	R V	25	9	N 70 E	R V
	1	N 19 E	R V	26	1	N 70 E	R V
12	1	N 86 W	R V	27	2	N 70 E	R V
13	1	N 27 E	Y	28	1	N 70 E	R V
	1	N 27 E	Y				

¹Shown on figure 37.

²Types of fracture-filling materials—R, red paint; O, orange paint; G, green paint; S, sand; C, cement.

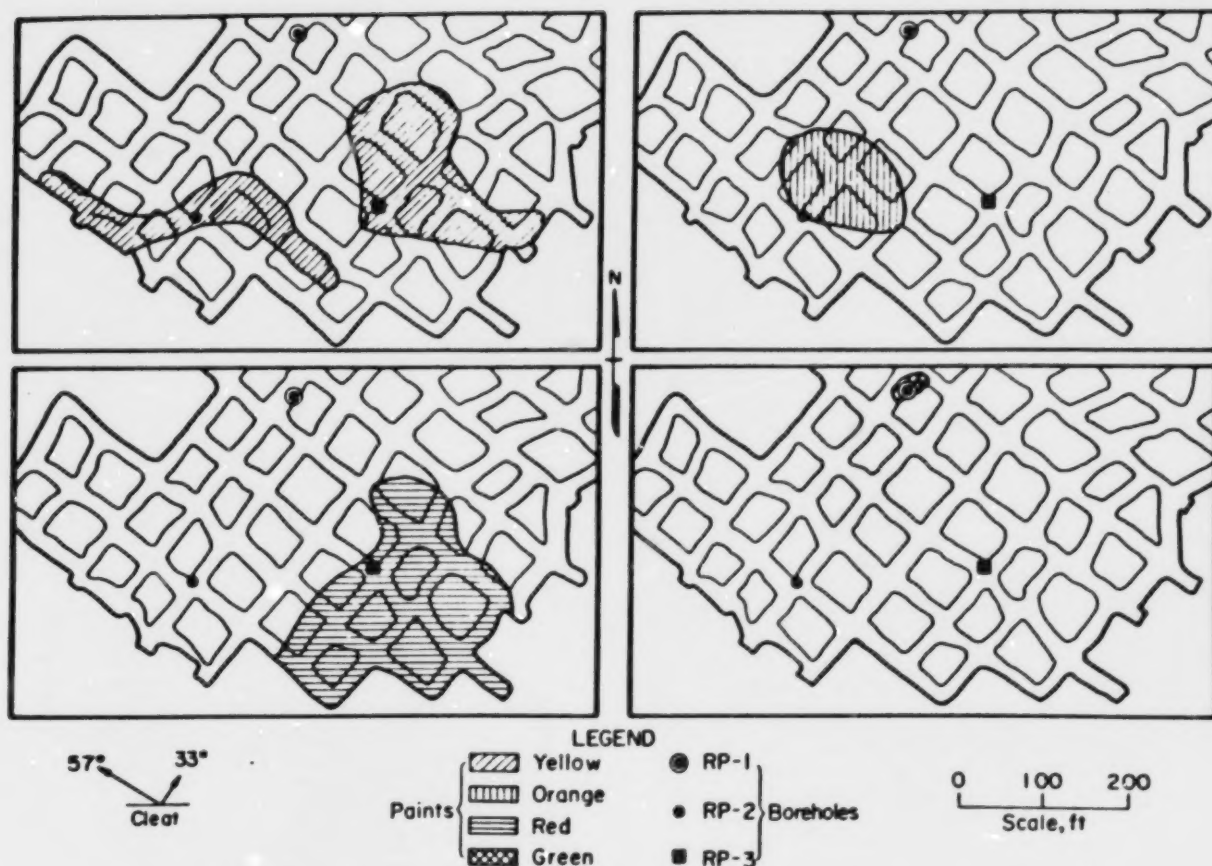


Figure 39.—Mine map, boreholes RP-1 through RP-3, with horizontal distribution of paint on roof (12).

LUCERNE NO. 6 MINE, BOREHOLE RP-2 (12)

Borehole Completion

Borehole RP-2 was drilled with air, 6 in. in diameter, to a depth of 660 ft, approximately 24 ft below the base of the target Upper Freeport Coalbed. The coalbed was 5.75 ft thick with a 7-in-thick shale parting near the top of the coal interval, and was located from 630.4 to 636.15 ft in the borehole. Prior to casing the borehole, sand was used to fill the hole to the top of the coalbed as an extra precaution against cement penetration of the coal interval. The borehole was cased with 4½-in.-OD casing with a cement basket shoe on the bottom as the primary means of keeping cement from reaching the coalbed below the casing. The bottom of casing was set at 627 ft, 3.4 ft above the top of the coalbed. Debris falling on top of the sand pack during the running of the casing prevented placing the bottom of the casing closer to the desired top of the coalbed. The cement basket shoe and sand pack were drilled out to a depth of about 2 ft below the base of the coalbed in preparation for stimulation.

Treatment Implementation (January 27, 1982)

Prestimulation strata characterization tests on this borehole were limited to gamma ray and density wireline

geophysical logs and laboratory mechanical property tests on general mine samples as described in the discussion of borehole RP-1. Prior to initiating the stimulation treatment, a jetting tool was positioned approximately 1.5 ft above the base of the coalbed and rotated to erode a horizontal slot in the coalbed below the shale parting. The stimulation treatment for borehole RP-2 was designed to consist of 30,000 gal of 75-pct-quality nitrogen-generated foam and 15,000 lb of 20/50-mesh sand proppant.

Breakdown pressures in borehole RP-2 were 1,170 lb/in²(ga) bottom hole and 945 lb/in²(ga) surface, using clear water. The continuous stimulation treatment record is shown in figure 40. Blue fluorescent paint was added to the foam pad. A summary of the paint colors, sand rates, and fluid volumes used in the treatment is given in table 7. Pump problems and time restrictions placed on the stimulation by the mine operator precluded pumping the scheduled 20,000-gal foam pad, which was reduced to 10,000 gal.

Treatment pressures declined from 1,210 to 1,105 lb/in²(ga) bottom hole and 1,150 to 1,060 lb/in²(ga) surface before problems with the fracturing pumps began. The initial sand concentration was 0.5 lb/gal of foam with orange fluorescent paint. After 13 min the sand concentration was increased to 1.0 lb/gal of foam with yellow fluorescent paint. The treatment pressure began a gradual increase almost immediately after increasing the sand

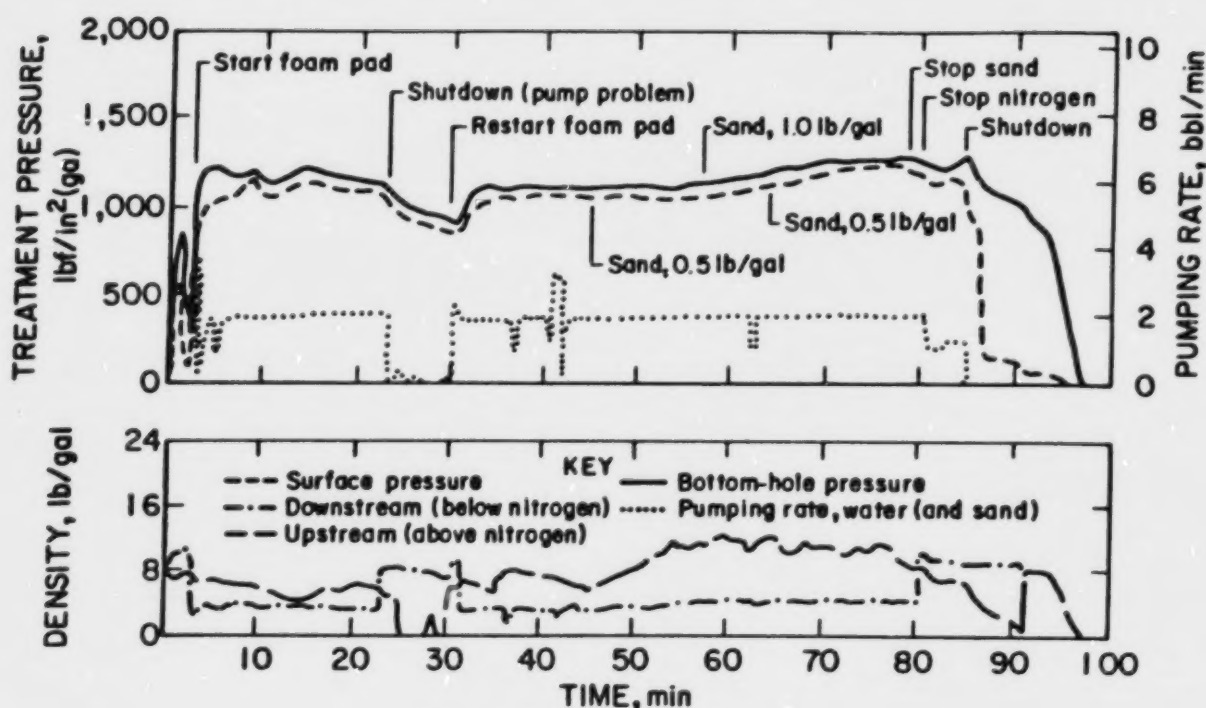


Figure 40.—Treatment record, borehole RP-2.

concentration, indicating either the inability of the induced fractures to accept, or inability of the foam to carry, the higher sand load. If the pressure had continued to increase unchecked, a point might have been reached where the foam could not have carried the suspended sand load, and a screenout or dropping of the sand in the wellbore would have occurred, prematurely terminating the treatment.

When it appeared that the bottom hole pressure would not stabilize or decline, the sand concentration was reduced to 0.5 lb/gal of foam for the remainder of the treatment. Maximum pressures during stage 2 were 1,300 lb/in²(ga) bottom hole and 1,250 lb/in²(ga) surface. The average surface treatment pressure was about 1,150 lb/in²(ga). A total of 9,800 gal of foam and 7,000 lb of sand were injected into the coalbed during the second stage of the treatment.

Underground Observations

Borehole RP-2 was mined through on March 25, 1982, approximately 2 months after stimulation. The location of the borehole was approximately 25 ft from the expected location plotted from a deviation survey of the borehole. Numerous vertical fractures were observed surrounding the borehole. As mining approached the borehole from the northeast (subparallel to the butt cleat), several fractures in the butt cleat direction (N 31° E) were intercepted head on and traced to the borehole (4) (fig. 37). One of the butt cleat fractures (26) was sand filled to the point where it entered a pillar, 30 ft from the borehole. The trace of this fracture where it emerged on the opposite side of the pillar (3), 67 ft from the borehole, contained only paint. These

fractures generally terminated at the floor and extended upward to the top of the shale parting. A horizontal component was also observed on the interface between the top of the shale parting and the upper coal bench. No extensions of the butt cleat fractures were observed beyond the borehole to the southwest.

Vertical fractures in the face cleat direction were observed as the borehole was mined through. One fracture (7) on the northwest rib (fig. 41) and one (5) to the southeast contained sand. The fracture on the northwest rib was closest to the borehole and was approximately 1/4 in wide and filled with sand proppant to the bottom of the parting. A second vertical fracture split off the main fracture approximately midway in the main bench of coal and rejoined near the base. Yellow and orange paint were found in the sand. The upper section of the fracture through the shale parting was filled with grout tagged with pink paint used to plug the hole for abandonment. The grout also extended a few inches horizontally along the interface between the shale parting and the upper bench of coal (fig. 41). Sand was found beyond the grout on the same horizontal plane in the rib for a maximum distance of 17 ft from the borehole to the northeast (fig. 42). The vertical sand-filled portion of the fracture narrowed to approximately 1/16 to 1/8 in where it extended to the southeast rib (5). Grout from the plugging operation was again found in the vertical fracture through the shale parting. Sand was present for a short distance horizontally at the interface between the top of the shale parting and the upper coal bench. A summary of the vertical fracture observations is given in table 8.

The extent of paint on the horizontal interface between the top of the shale parting and the upper coal

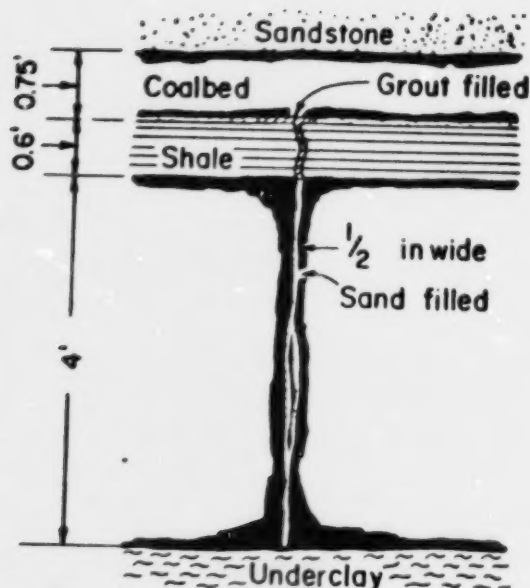


Figure 41.—Cross section of fractures at location 7 on the northwest rib near borehole RP-2. (Modified from Murrie (12))

bench (fig. 42) indicated a wide distribution of fluids on that surface. The orange paint was found a maximum of 265 ft from the wellbore. The yellow paint from RP-2 and the nearby RP-3 apparently overlapped, but the maximum horizontal extent seems to have been about 200 ft. Yellow and orange paint were also found on the horizontal plane between the top of the coal and the roof. The horizontal extent of paint on this plane, as shown in figure 39, was substantially less than on top of the shale parting. The pattern of the yellow paint on the roof, with two lobes elongated parallel to the face cleat and a connecting lobe parallel to the butt cleat suggests some influence of the vertical coal cleat on the horizontal distribution of the stimulation fluids. No sand was found along the interface between the roof and coalbed. Most of the sand proppant injected into this borehole was contained in only a few vertical fractures. There was no evidence of roof penetration by the stimulation fluids.

LUCERNE NO. 6 MINE, BOREHOLE RP-3 (12)

Borehole Completion

Borehole RP-3 was drilled with air, 6-in. in diameter, to a depth of 675 ft, approximately 35 ft below the base of the target Upper Freeport Coalbed. The coalbed was 5.5 ft thick with a 5-in-thick shale parting near the top of the



Figure 42.—Mine map, boreholes RP-1 through RP-3, with horizontal distribution of paint and sand on top of shale parting (12).

coal interval and was located between 634.6 and 640.1 ft in the borehole. Prior to casing the borehole, sand was used to fill the hole to the top of the coalbed as an extra precaution against cement penetrating the coal interval. The borehole was cased with 4½-in.-OD casing with a cement basket shoe on the bottom as the primary means of keeping cement from reaching the coalbed below the casing. The bottom of casing was set at 632 ft, 2.6 ft above the top of the coalbed. As was the experience at the other two boreholes at this mine site, debris on top of the sand pack prevented placing the bottom of casing closer to the desired top of the coalbed. The cement basket shoe and sand pack were drilled out to a depth of about 2 ft below the base of the coalbed in preparation for stimulation.

Treatment Implementation (January 30, 1982)

Prestimulation strata characterization tests on this borehole were limited to natural gamma ray and density wireline geophysical logs and laboratory mechanical property tests on general mine samples as described in the discussion of borehole RP-1. Prior to initiating the stimulation treatment, a jetting tool was positioned approximately 1.5 ft above the base of the coalbed and rotated to erode a horizontal slot in the coalbed below the shale parting. The stimulation treatment for borehole RP-3 was designed to consist of 30,000 gal of 75-pct-quality nitrogen-generated foam and 15,000 lb of the 20/50-mesh sand proppant.

The stimulation of borehole RP-3 began with a small volume of clear water to break down the coalbed. The continuous stimulation treatment record is shown in figure 43. Breakdown pressures were 1,320 lb/in²(ga) bottom hole and 1,080 lb/in²(ga) surface. A 20,000-gal foam pad with red fluorescent paint was injected at a rate of 6 bbl/min. At the end of the pad, sand proppant was added to the treatment in concentrations that were gradually increased from 0.5 to 1.5 lb/gal of foam in increments of 0.5 lb/gal. At each increase in sand concentration a different paint color was introduced as shown in table 7. A total of 11,500 lb of sand was injected with 10,200 gal of foam during stage 2. Pressures declined from 1,365 lb/in²(ga) bottom hole and 1,110 lb/in²(ga) surface at the beginning of the foam pad to 815 lb/in²(ga) bottom hole and 750 lb/in²(ga) surface at the end of the treatment with no apparent increase in pressures as sand was introduced or its concentration increased. Average surface treatment pressure was about 850 lb/in²(ga).

During the latter portion of the stimulation treatment of borehole RP-3, communication was observed with borehole RP-2, where small amounts of foam were detected bubbling from the casing. The two boreholes are approximately 225 ft apart and on a line that is oriented 25° from the face cleat direction. It is possible that the decrease in treatment pressure resulted in part from the fractures created in the stimulation of borehole RP-3 entering the area previously affected by the treatment of borehole RP-2.

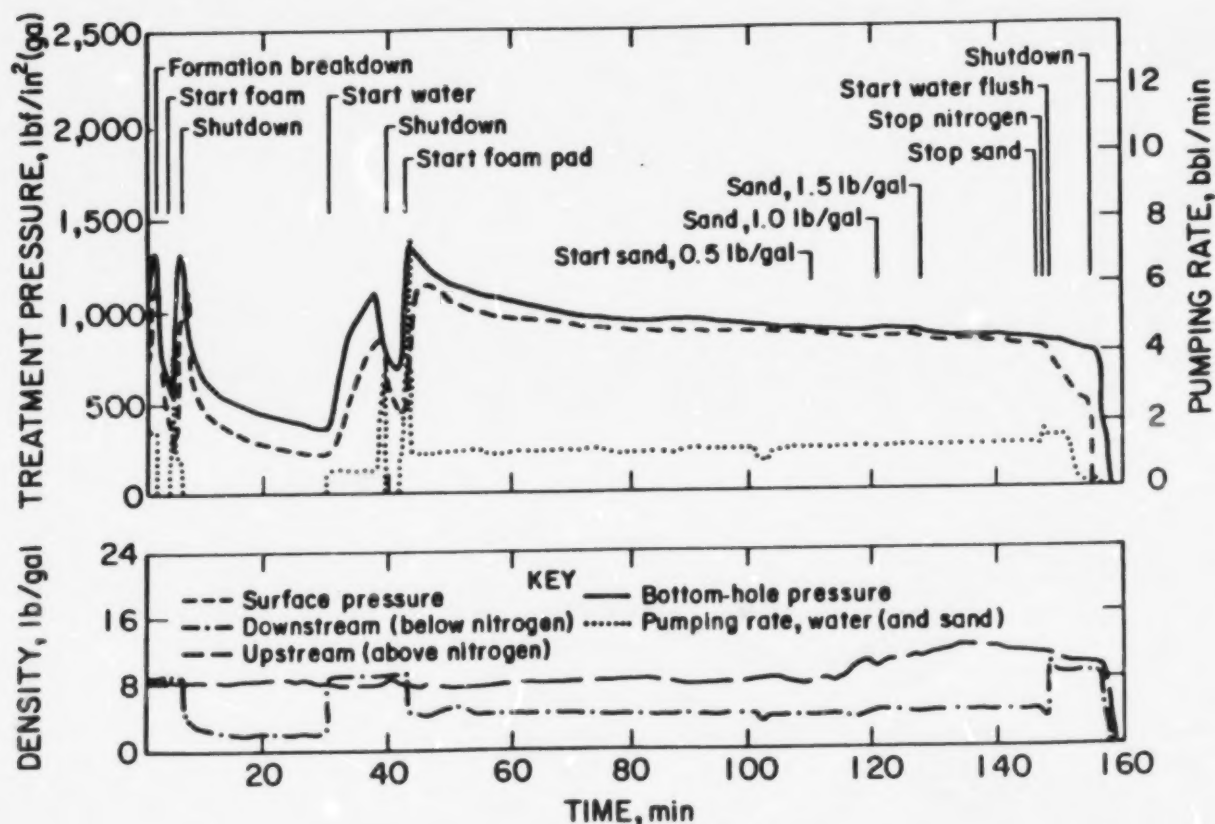


Figure 43.—Treatment record, borehole RP-3.

Underground Observations

Borehole RP-3 was mined through on May 18, 1982, approximately 4 months after stimulation. The location of the borehole was approximately 25 ft from the expected location plotted from a deviation survey of the borehole. The greatest number of vertical fractures at the Lucerne No. 6 Mine site were found associated with borehole RP-3 (fig. 37), which also had the largest stimulation treatment. Most of the vertical fractures seen near the wellbore were parallel to the face cleat. One butt cleat fracture was traced from the borehole (19) to the rib (22) of the pillar south of the hole and several others (12-13, 20-21) were observed at greater distances away from the borehole. The only sand-filled fractures (15-16) were in the face cleat direction and were oriented in a direction that would intercept the borehole. Representative cross sections of the northwest (16) and southeast (15) wings of this dominant fracture are shown in figure 44. The other vertical fractures associated with this borehole contained various combinations of red, yellow, and orange fluorescent paint on cleat surfaces in the lower bench of coal.

The vertical fracture on the northwest rib (16) was filled with sand, had a maximum width of approximately 1 in, and tapered to the width of a cleat at the base. This sand-filled fracture extended upward through the shale parting and became horizontal on top of the parting. Orange and yellow paint were found in the fracture (16) nearest the hole in the northwest direction and only red was found further out (23) in this direction. The sand-filled wing of this major fracture (15) on the southeast rib (fig. 44) was slightly more complex than that on the northwest rib. Two nearly parallel vertical sand- and grout-filled (from the plugging operation) fractures were present. The sand filling extended from the floor to about halfway up the main coal bench, and the grout filled

the remainder of each fracture up to its maximum vertical extent. Maximum width was $\frac{1}{2}$ in. The shorter vertical fracture terminated approximately 2 in below the parting. The other fracture terminated at the bottom of the shale parting, but a horizontal sand-filled fracture split off from it 2 in below the parting. This split intercepted the parting at a shallow angle and then cut through it at approximately 45° and became horizontal on top of the parting. Orange and yellow fluorescent paint were found in the fractures (15) closest to the borehole in this direction, red only slightly further away (24), and finally yellow only at the furthest extent of this fracture on the next rib (28). A summary of the vertical fracture observations is given in table 8.

Sand on the interface between the top of the shale parting and the upper bench of coal was found a maximum of 200 ft from the borehole (fig. 42). Yellow was the most widespread paint on top of the binder, extending a maximum of 200 ft, however, there was an apparent overlap with the yellow paint injected in borehole RP-2. Red paint was found over a wide area, and was distributed in a pattern that consisted of several lobes elongated parallel to the face and butt cleat directions. This again suggests some control of the horizontal fluid placement by the cleat system in the coalbed. Very little orange paint was found on the top of the shale parting, perhaps because it was masked by the red and yellow paints. No blue paint was seen in any of the fractures at the Lucerne No. 6 Mine, probably because of the inability to distinguish it from the glow of the ultraviolet light being used and because of large areas of blue fluorescent hydraulic fluid (from leaking hoses on mining equipment) on the ribs. No sand was found on the interface between the roof and the top of the coalbed, but red and yellow fluorescent paint were mapped as shown in figure 39.

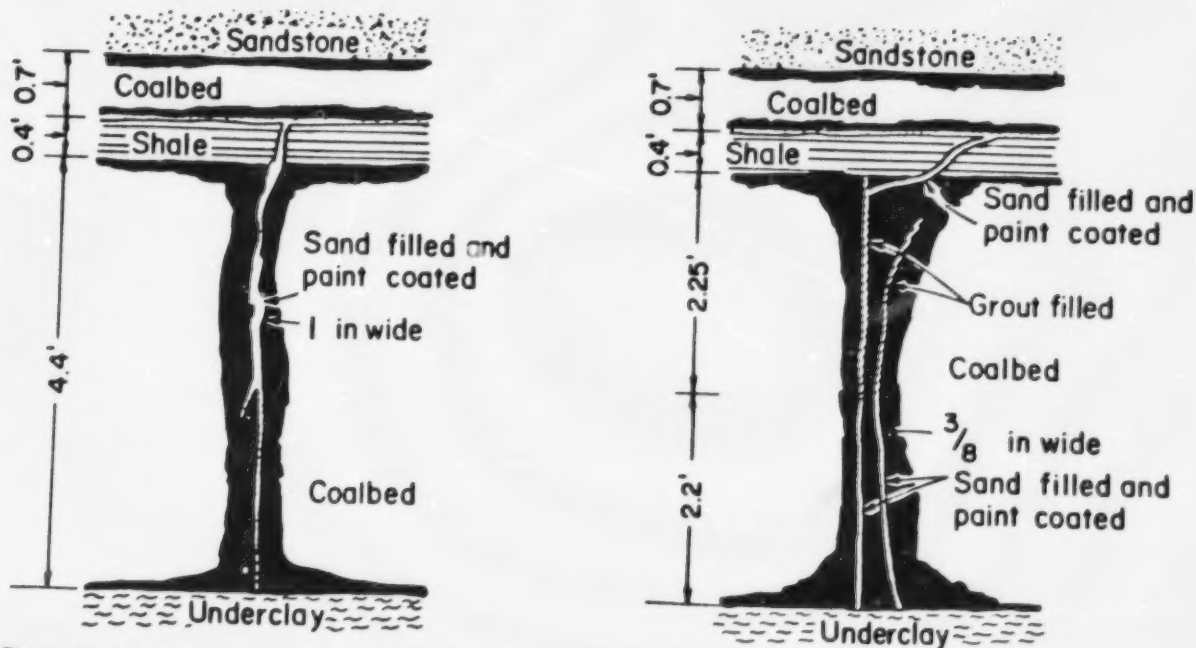


Figure 44.—Cross section of fractures at location 15 on southeast rib (right) and location 16 on northwest rib (left) near borehole RP-3. (Modified from Murrie (17))

LOWER KITTANNING COALBED, WEST VIRGINIA

KIT NO. 1 MINE, BOREHOLE KE-2 (12)

Borehole Completion

Borehole KE-2 was drilled with air, 7 in. in diameter, to a depth of 685 ft, approximately 20 ft below the base of the target Lower Kittanning Coalbed. The coalbed was 5.2 ft thick, and was located at a depth of 660 to 665.2 ft in the borehole. Prior to casing the borehole, sand was used to fill the hole to the top of the coalbed as an extra precaution against cement penetration of the coal interval. The borehole was cased with 4½-in.-OD casing with a cement basket shoe on the bottom as the primary means of keeping cement from reaching the coalbed below the casing. The bottom of casing was set at 659.5 ft, 0.5 ft above the top of the coalbed.

Treatment Implementation (March 19, 1982)

Prestimulation strata characterization tests on this borehole included caliper, gamma ray, and density wireline geophysical logs. A series of mechanical property tests to measure triaxial compressive strength, Young's modulus, and residual friction angles and coefficients were conducted on coal and surrounding rock samples collected at the mine. Analysis of the test results indicated

that the roof shale tested was abnormally weak and soft for the rock type (25). This weak roof material is probably at least partially responsible for the severe roof control problems generally experienced at the mine. Extensive mapping of existing geologic conditions was conducted at the mine to aid in interpreting the effect of the treatment on the coalbed and surrounding strata. The details of the geologic characterization can be found in reference 12.

Borehole KE-2 was stimulated with 75-pct-quality nitrogen-generated foam and sand proppant. The continuous stimulation treatment record is shown in figure 45. The coalbed was broken down using a small volume of water. Pressures were 1,050 lb/in²(ga) bottom hole and 1,000 lb/in²(ga) surface. The 20,000-gal foam pad with red fluorescent paint was injected into the coalbed at a rate of 8 bbl/min. Treatment pressures gradually increased from 1,050 to 1,130 lb/in²(ga) bottom hole and 1,000 to 1,090 lb/in²(ga) surface at the end of the foam pad. Stage 2 began with the introduction of sand proppant at a concentration of 1.5 lb/gal along with the yellow fluorescent paint. Treatment pressure increased to 1,340 lb/in²(ga) bottom hole and 1,290 lb/in²(ga) surface, leveled off, then rapidly rose to 2,300 lb/in²(ga) bottom hole and 2,000 lb/in²(ga) surface. The rise in treatment pressure was so rapid that the sand injection could not be stopped quickly enough to prevent a screenout and

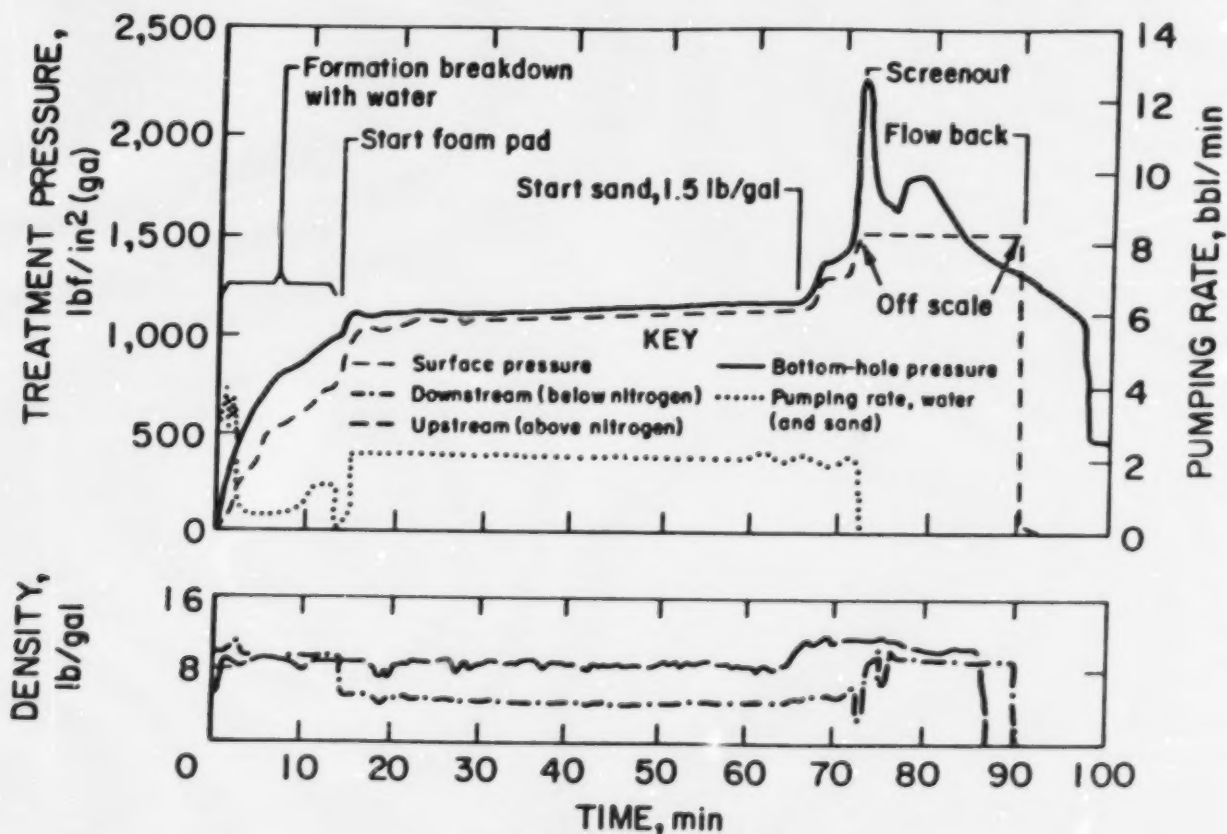


Figure 45.—Treatment record, borehole KE-2.

termination of the treatment. Only 3,500 gal of the scheduled 7,000 gal of foam and 3,300 lb of the scheduled 10,000 lb of sand proppant were injected during stage 2. A review of the treatment record after the job indicated that the calculated foam quality was 54 to 60 pct, not the required 75-pct quality necessary to carry the sand concentration specified, thereby contributing to the screenout.

Underground Observations

A paint-coated fracture from borehole KE-2 was intercepted on November 18, 1982, approximately 8 months after stimulation. The stimulation treatment from borehole KE-2 was intercepted at a point approximately 85 ft from the borehole (fig. 46). A single vertical face cleat plane with red paint was observed. No sand proppant was found. The paint terminated at the floor and disappeared before reaching the roof. A small area of red paint was found on the roof shale approximately 38 ft from the vertical fracture, but none was observed on the sandstone roof exposure closer to the vertical fracture. Poor roof conditions were found in the transition zone between the shale and sandstone roof; however, similar roof conditions are common throughout the Kitt No. 1 Mine. Because of the poor roof conditions generally associated with this type of transition zone, several longwall panels were shortened to avoid mining through the zone and under the sandstone roof rock. Because of the change in the mining plans, the underground characterization of the stimulation was limited. There was no evidence of any penetration of the roof by the stimulation treatment nor any evidence of any influence on the roof conditions.

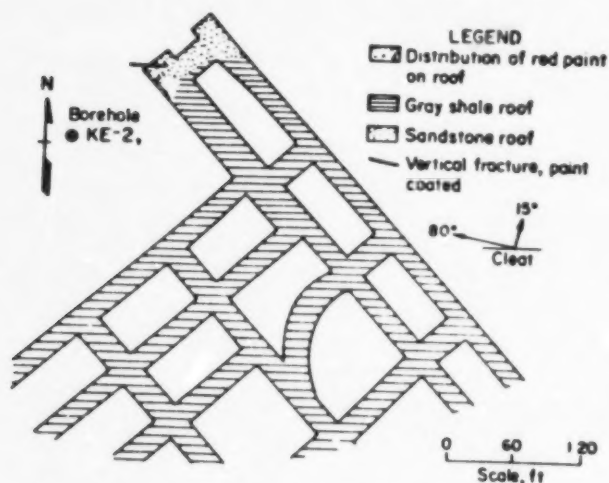


Figure 46.—Mine map, borehole KE-2, with fracture orientation. (Modified from Murrie (12))

KITT NO. 1 MINE, BOREHOLE DGBH-5

Borehole Completion

Borehole DGBH-5 was drilled with air, 10 in. in diameter, to a depth of 653.5 ft, approximately 11 ft below the base of the target Lower Kittanning Coalbed. The coalbed was 5.5 ft thick, and was located at a depth of 637.3 to 642.8 ft in the borehole. Prior to casing the borehole, sand was used to fill the hole to the top of the coalbed as an extra precaution against cement penetration of the coal interval. The borehole was cased with 7-in.-OD casing with a cement basket shoe on the bottom as the primary means of keeping cement from reaching the coalbed below the casing. The bottom of casing was set just above the top of the Lower Kittanning Coalbed.

Treatment Implementation (April 2, 1982)

Prestimulation strata characterization tests on this borehole included caliper, gamma ray, and density wireline geophysical logs. The treatment for borehole DGBH-5 consisted of two stages, the first being a 40,320-gal foam pad and the second 10,080 gal of foam with 11,800 lb of 20/50-mesh sand proppant. The treatment was conducted through the annulus between the casing and the tubing used to obtain bottom hole pressure. The coalbed was first broken down with a small volume of clear water at pressures of 860 lbf/in²(ga) bottom hole and 1,210 lbf/in²(ga) surface. A continuous stimulation record for the treatment is shown in figure 47. Treatment pressures gradually declined to 490 lbf/in²(ga) bottom and 1,140 lbf/in²(ga) surface at injection rates of 8 bbl/min. Average surface treatment pressure was about 950 lbf/in²(ga). Sand proppant was introduced at the end of the foam pad in concentrations that were gradually increased from 0.25 to 1.5 lb/gal in increments of 0.25 lb/gal. Because of mechanical problems with the methane engines used to power the pumping unit on this borehole, continuous water and gas production were never achieved.

Underground Observations

Sand-filled fractures from borehole DGBH-5 were intercepted by mining in August 1984, approximately 2½ yr after stimulation. A sand-filled vertical fracture was intercepted in the entry closest to the borehole (fig. 48). The fracture was approximately 1½ in wide on the east rib closest to the borehole. The fracture paralleled the face cleat, but followed a sinuous path as shown in figure 49. The fracture extended from the floor to the roof, but did not penetrate the roof. The mine operator reported a flow of water and gas from the fracture for some period of time after interception. The flow of water and gas was probably due to the inability to produce the borehole prior to interception.

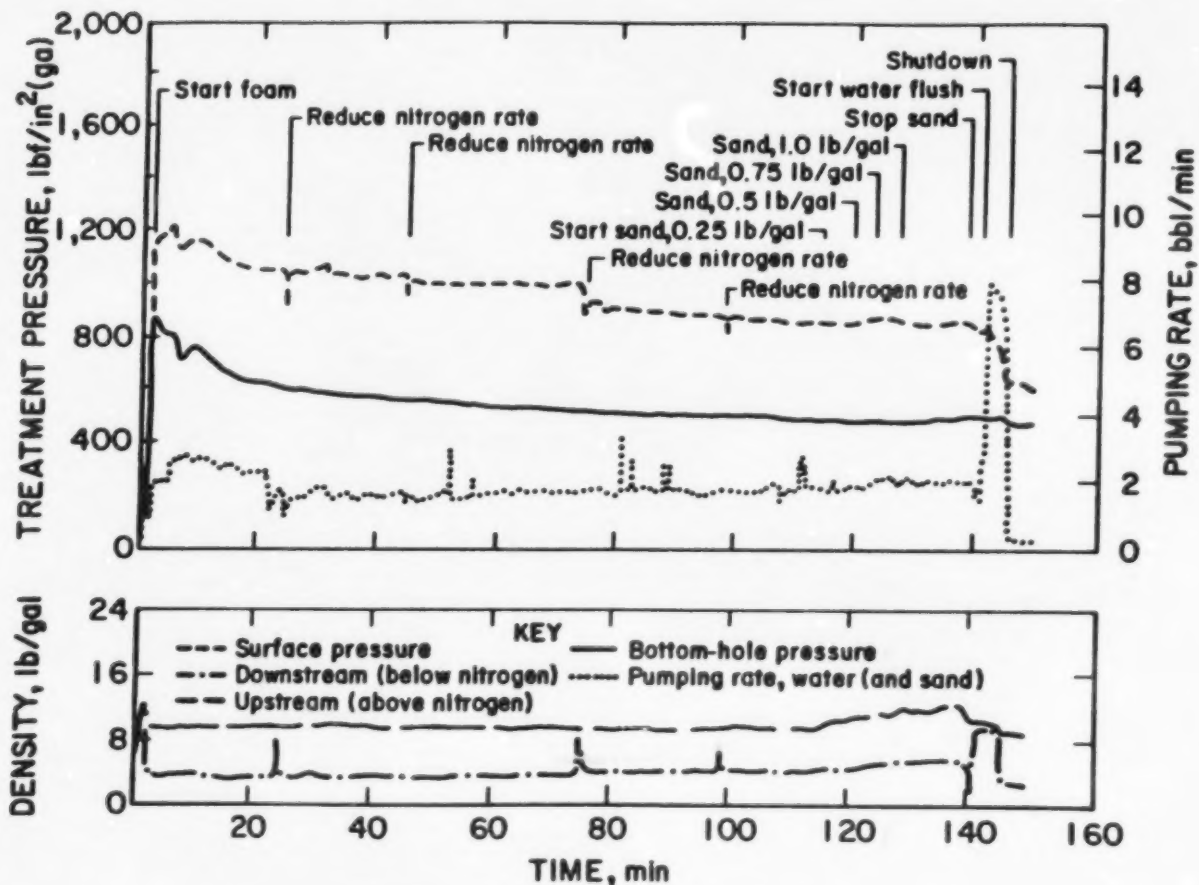


Figure 47.—Treatment record, borehole DGBH-5.

The fracture became horizontal at the interface between the top of the coalbed and the roof where it was generally only a few sand grains thick. Sand could be traced horizontally along the east rib a maximum of 29 ft from the vertical fracture.

The vertical fracture was also found on the opposite side of the entry on the west rib. The fracture was $\frac{1}{2}$ in wide and extended from floor to roof (fig. 49). The fracture was parallel to the face cleat along a sinuous path similar to that on the east rib. There were several thin (several sand grains wide) vertical fractures in the lower section of the coalbed parallel to the primary fracture. The fracture became horizontal at the interface between the top of the coalbed and the roof, and sand could be traced a maximum of 51 ft from the position of the vertical fracture on the west rib. The distance from the presumed location of the borehole to the vertical trace on the west rib was 72 ft and the maximum distance that sand on the horizontal surface was found from the presumed location of the borehole was 105 ft. No penetration of the roof was observed at the west rib or across the roof of the entry between the two vertical exposures. A thorough inspection of the coalbed on both sides of the entry on the opposite side of the west pillar failed to reveal any sand-filled vertical or horizontal extension of the fracture.

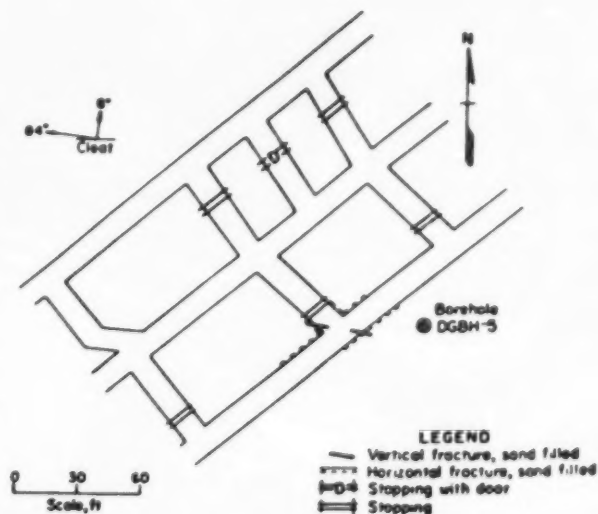


Figure 48.—Mine map, borehole DGBH-5, with fracture orientations.

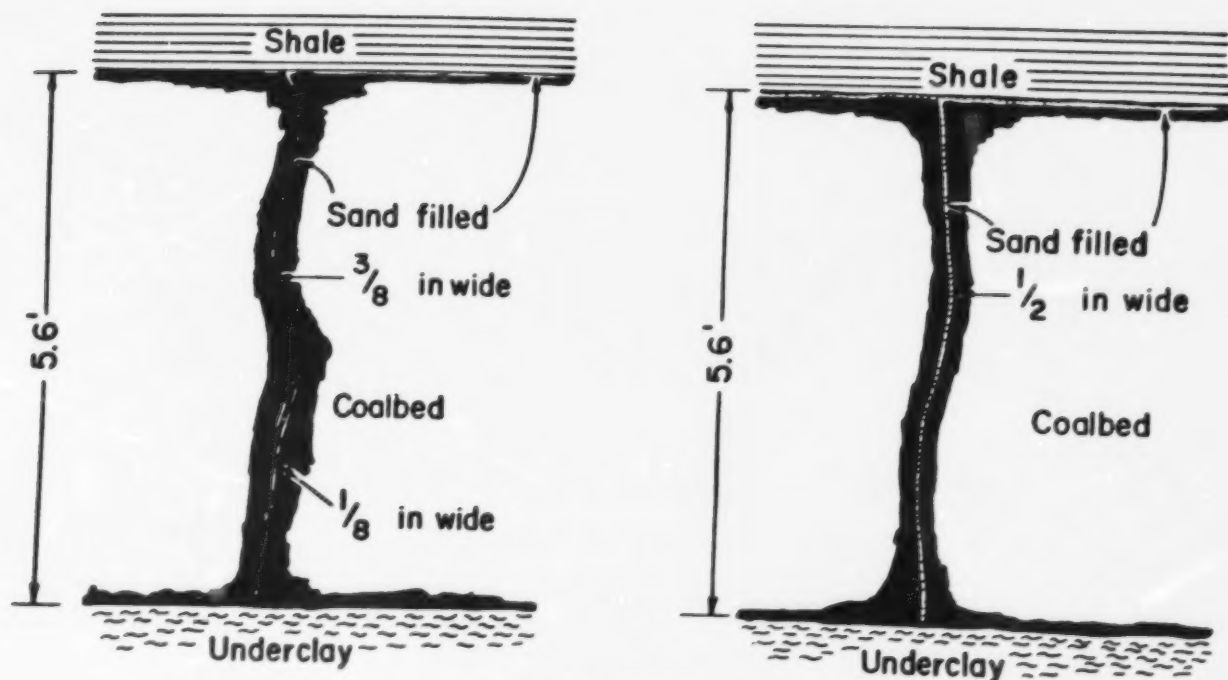


Figure 49.—Cross section of fractures on east (right) and west (left) ribs near borehole DGBH-5.

ILLINOIS NO. 6 COALBED, ILLINOIS

INLAND STEEL MINE, BOREHOLE 1-NE (8)

Borehole Completion

Borehole 1-NE was drilled 9 in. in diameter to a depth of 743 ft. The hole was cased to within a few feet of the top of the coalbed with 7-in-OD casing. The target Illinois No. 6 Coalbed was 9 ft thick and was located from approximately 729 to 738 ft in the borehole. The coalbed was overlain by a thick sandy shale.

Treatment Implementation (February 19, 1974)

Borehole 1-NE was treated through 2½-in-diameter tubing with a packer set in the casing at a depth of 720 ft, and the open end of the tubing extending to the middle of the coalbed at a depth of 733 ft. The treatment design to be implemented was to use gelled water (20 lb of guar gum per 1,000 gal of water) and sand to create two fracture wings, each 431 ft long, of which 305 ft would be propped by sand, 6.9 ft high and 0.2 in wide. A 2,000-gal gelled water pad was first injected into the coalbed to control leakoff. The continuous stimulation treatment record is shown on figure 50. Formation break was reported at 1,050 lb/in²(ga) surface. A pumping rate of 10 bbl/min was maintained throughout the treatment. Following the pad, an additional 10,000 gal of gelled water and 6,400 lb of 10/20-mesh sand was injected into the coalbed according to the schedule shown in table 9. The average injection pressure was 850 lb/in²(ga) surface. Gas flows during the

short production life of the borehole reached only 4,300 ft³/d, due in part to the low gas content of the coalbed.

Underground Observations

Fractures from the treatment were intercepted by mining in May 1974, approximately 3 months after stimulation. A single sand-filled vertical fracture was observed in four entries northeast of the presumed location of the borehole (fig. 51). The fracture extended from the roof to a hard shale parting approximately 2 ft from the bottom of the 9-ft-thick coalbed. The width of the fracture ranged from ¼ to ½ in, averaging approximately ¼ in. A small amount of water drained from the fracture at the time of the mine-through. The sand-filled fracture could be traced across several entries in the 1 ft of roof coal generally left to protect the overlying shale from deterioration. The fracture in the roof coal was essentially the same width as the vertical exposures on the rib.

The orientation of the vertical fracture (N 76° E) did not coincide with either the face (N 34° W) or butt (N 45° E) cleat, but was closest to the butt cleat orientation (fig. 51). It is possible that the induced fracture orientation was influenced by the horizontal stress field in the area. The major horizontal stress axis in south-central Illinois has generally been reported (26-27) to be east to east-northeast, which would correspond quite well to the N 76° E fracture orientation. The reported fracture orientation could also be the result of unobserved stairstepping along face and butt cleat, which could give an apparent orientation for the fracture(s) oblique to the cleat orientation.

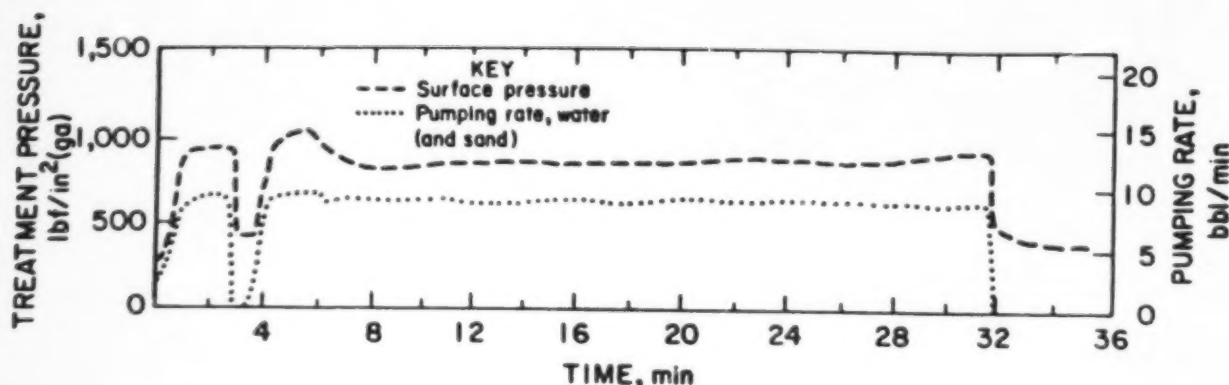


Figure 50.—Treatment record, borehole 1-NE. (Modified from Elder (8))

Table 9.—Summary of treatment chronology, borehole 1-NE (8)

Fluid volume, gal	Sand rate, lb gal
2,000 (pad)	NAP
3,000	0.25
2,000	50
1,800	75
3,200	1
NAP: Not applicable.	

At its maximum lateral extent, the vertical fracture was observed 416 ft from the presumed location of the borehole. This is the greatest length observed to date for a sand-filled vertical fracture wing. There was essentially no penetration of the roof rock except at one point where Elder (8) reported "there was some evidence of fluid and sand intruding a few inches into a rock joint that crossed the induced fracture." No adverse mining conditions were reported as a result of the stimulation treatment, and even at a follow-up visit 2.5 yr after the stimulation there was still no deterioration. No horizontal fracture component was noted at this mine site.

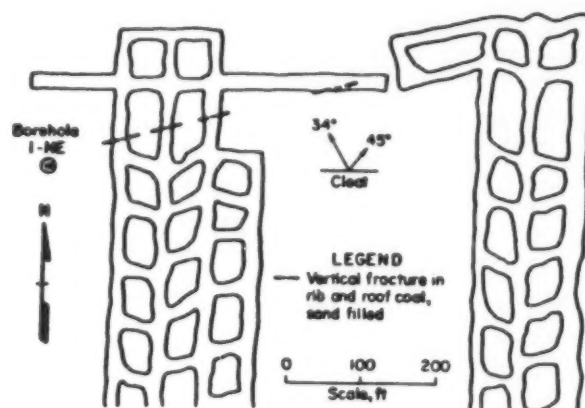


Figure 51.—Mine map, borehole 1-NE, with fracture orientations. (Modified from Elder (8))

JAWBONE COALBED, VIRGINIA

McCLURE NO. 1 MINE, BOREHOLE DG-1A (9)

Borehole Completion

Borehole DG-1A was drilled to a depth of approximately 455 ft. The hole was cased with 4½-in.-OD casing to within about 2 ft of the top of the target Jawbone Coalbed. A packer shoe on the bottom of casing was used to keep cement from reaching the coalbed. The coalbed was 5 ft thick and was located from 425 to 430 ft in the borehole.

Treatment Implementation (July 29, 1978)

The borehole was treated down the casing with a packer installed at the surface. A continuous treatment record is not available for this borehole. The treatment was initiated with a 4,200-gal water pad. A formation break was reported at 1,000 lb/in²(ga) surface. The

primary treatment consisted of 35,300 gal of 75-pct-quality nitrogen-generated foam and 28,000 lb of 20/40-mesh sand proppant. The average pumping rate was 16 bbl/min. During a continuous production period over the first half of 1979, the borehole reached a maximum gas production rate of 45,000 ft³/d, then gradually declined to slightly less than 30,000 ft³/d. A detailed production history of the borehole can be found in the paper by Lambert (9). Chronic mechanical problems with the methane engine used to power the water pumping unit severely limited the production potential of the borehole.

Underground Observations

Borehole DG-1A was mined through on May 12, 1980, approximately 2 yr after stimulation. Both vertical and horizontal sand-filled fractures were observed, with the horizontal fractures predominating at the shallow depth

of the coalbed. Vertical sand-filled fractures were found a maximum of 20 ft from the borehole and were oriented parallel to the face cleat. The vertical fractures were completely contained in the coalbed, and ranged from $\frac{1}{8}$ in to a hairline in width. No evidence of roof penetration was observed. The horizontal fractures were generally

observed approximately 2 to $5\frac{1}{2}$ in below the top of the coalbed and ranged in thickness from $\frac{1}{8}$ in to a hairline. The sand-filled horizontal fractures were observed over a large, generally elliptical, area around the borehole. The sand-filled horizontal fractures were observed a maximum of about 250 ft from the borehole.

ROCK CANYON COALBED, UTAH

SOLDIER CANYON MINE, BOREHOLE SC-1

Borehole Completion (12)

Borehole SC-1 was drilled 6 in. in diameter to a depth of 740 ft, approximately 42 ft below the base of the target Rock Canyon Coalbed. The borehole was cased with $4\frac{1}{2}$ -in.-OD casing to the top of the coalbed. A cement basket shoe on the bottom of casing was used in addition to filling the hole with sand to the top of the coalbed to keep the cement from reaching the coalbed. The Rock Canyon Coalbed was 13.8 ft thick with 1.7 ft of shale partings in the upper 3.2 ft of the coalbed. The coalbed was located at a depth of 684 to 697.8 ft in the borehole.

Treatment Implementation (August 5, 1982) (12)

Prestimulation strata characterization included gamma ray, density, and caliper wireline geophysical logs. A jetting tool was positioned about 7 ft above the base of the coalbed and rotated to cut a horizontal slot. Coal particles

in the fluid returns indicated that the tool had been placed in the coal interval. The stimulation treatment utilized 75-pct-quality nitrogen-generated foam, sand proppant, and green fluorescent paint. A 30,000-gal foam pad was injected first to control leakoff, followed by 20,000 gal of foam containing 15,000 lb of 20/40-mesh sand proppant. There was some evidence that the foam quality was lower than the specified 75 pct, therefore the actual foam volumes may be significantly less than reported by the service company. Foam treatment rates were maintained at approximately 16 bbl/min. Surface pressures during the injection of the foam pad quickly reached 1,000 $\text{lb/in}^2(\text{ga})$ then gradually increased to about 1,250 $\text{lb/in}^2(\text{ga})$. No obvious formation break was observed. The surface treatment pressure during sand injection reached a maximum of 1,500 $\text{lb/in}^2(\text{ga})$. The average surface treatment pressure was about 1,350 $\text{lb/in}^2(\text{ga})$. The continuous treatment record for this stimulation is shown in figure 52. The borehole was not completed for production since the project goals were to only stimulate and evaluate the effect of the stimulation treatment on the coalbed and surrounding strata.

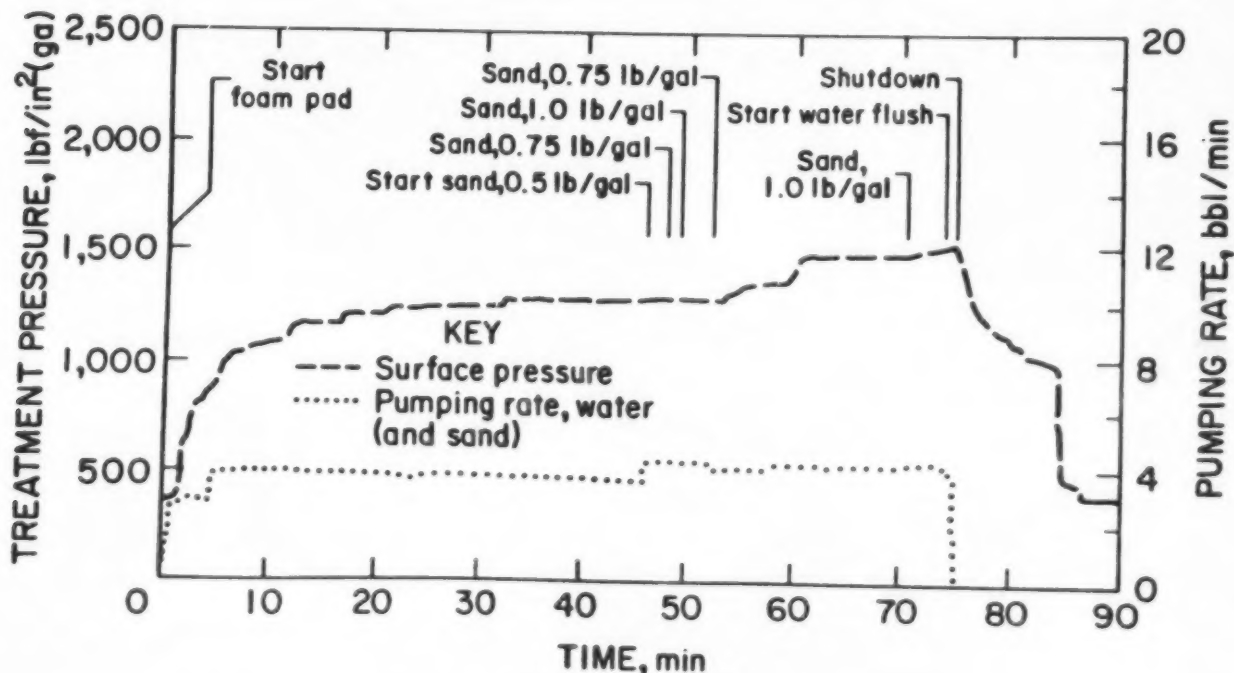


Figure 52.—Treatment record, borehole SC-1.

Underground Observations

The area around the presumed location of the borehole was mined through in April 1985. A directional survey run on the borehole was used to plot the coalbed intercept point, which was calculated to be approximately 35 ft north-northeast of the surface location and near the center of an entry (fig. 53). A thorough examination of the roof and ribs at the expected underground borehole location and all accessible entries in the area failed to reveal the borehole or any of the sand proppant used in the stimulation.

Examination of the area with the ultraviolet light revealed a substantial amount of green fluorescent material on the coal faces, and initially it was thought to be the green fluorescent paint used in the treatment. The fluorescence was generally restricted to short (less than 2-in) horizontal patches or bands in the coal. Closer examination revealed that the fluorescing material was actually naturally occurring resins that are common in the Cretaceous age coalbeds of the western United States. Blacklight surveys were then conducted at points as far as several thousand feet from the presumed location of the borehole, where there was essentially no possibility of the presence of paint from the stimulation. Green fluorescing material of the same character as was observed near the presumed borehole location was also seen at these remote locations.

There seemed to be more green fluorescence in the area of the presumed borehole location, so it is possible that some paint was actually present, but if so, it could not be distinguished from the naturally occurring fluorescent materials. It is also possible that there actually was more

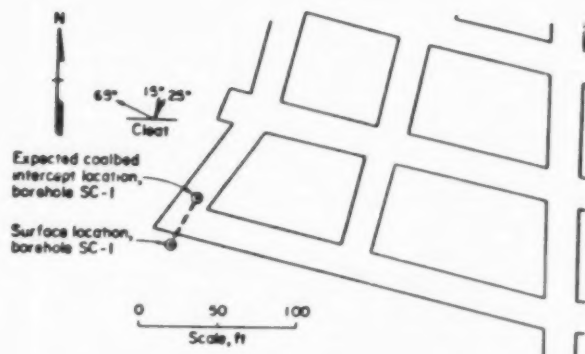


Figure 53.—Mine map, borehole SC-1.

naturally occurring fluorescent material at this location, or that it was more noticeable because of the fresh, unweathered exposures. In any case, it would be advisable to check for any naturally occurring fluorescence before selecting a paint color for use in a stimulation treatment.

If the borehole was located within the general area expected, and if the stimulation generally followed the face cleat orientation (N 65° W) as would be assumed, then a wing of the fracture would have a high probability of being intercepted by the available entries at the time of observation (fig. 53). The obvious incorrect presumed location of borehole SC-1 points out the need for accurate surveys (surface, underground, and borehole deviation) and the advantages of actually mining through the borehole in interpreting underground observations.

SUMMARY OF OBSERVATIONS

The areas affected by 22 Government-sponsored stimulation treatments in coalbeds have been mined through and observed underground. The underground observations have revealed a variety of conditions ranging from extensive vertical and horizontal sand-filled and/or paint-coated fractures to no discernible fractures. A summary of the treatment parameters and underground observations is given in tables 10 and 11. Eight of the treatments included the addition of fluorescent paint as an aid in locating and mapping the paths of fluid movement in the exposed coalbed and surrounding strata. With the paint, it was generally possible to trace the paths of fluid movements farther than the presence of the more obvious sand-filled fractures. It was also possible to identify additional zones of multiple fluid pathways. A good example of the additional traceable fracture length was observed at borehole DHM-6 (Oak Grove Mine, Alabama), where sand-filled fractures could be traced a maximum of 95 ft compared to paint-coated cleat observed 630 ft from the borehole.

When paint-coated fractures were identified using the ultraviolet light, the fracture width was essentially only the width of the naturally occurring cleat that had been penetrated. The cleat would not have been identified as a pathway for stimulation fluid without the presence of the paint. It is not known if the paint-coated cleat in fact represents a path of increased permeability that will enhance the flow of water and gas to the wellbore.

Most of the observed sand-filled and paint-coated vertical fractures penetrated or paralleled face cleat and occasionally butt cleat in the coalbed. It is interesting to note that the longest sand-filled vertical fracture (416 ft, borehole 1-NE, Illinois No. 6 Coalbed) did not parallel a cleat direction, but was approximately 30° from the butt cleat orientation. This may be the result of the influence of local horizontal stress fields or unobserved stairstepping, which gave an apparent orientation oblique to the cleat orientation. At several locations, fractures that paralleled the face cleat did not directly intercept the borehole if extended back from their point of observation. Apparently these treatment fluids followed a stairstep pathway through the coalbed, alternating along face and butt cleat, with only the face cleat fractures being observed on the exposed ribs and pillars.

Vertical sand-filled fractures were generally wider nearest the borehole and narrowed rapidly away from the borehole. The maximum lateral extent of sand-filled vertical fracture wings was generally short. Nine boreholes had sand-filled fracture wings with a maximum length of 30 ft or less, three boreholes had fracture wings 70 to 100 ft in length, and only four had wings over 100 ft in length. Zones of multiple, thin parallel sand-filled and/or paint-coated fractures are common near the boreholes. At increasing distances from the boreholes, the number of multiple vertical fractures usually decreases and only a single fracture remains. Vertical fractures also

Table 10.—Summary of stimulation treatment data

Borehole	Depth (top of coal), ft	Treatment	Fluid		Sand, lb	Surface pressure, lb/in ² (g)	
			Volume, gal	Injection rate, bbl/min		Average	Maximum
BLUE CREEK COALBED, OAK GROVE MINE, JEFFERSON COUNTY, AL							
TW-1	1,113.0	Gelled water	5,290	2 - 10.5	2,500	650	1,775
TW-2	1,093.4	do	3,500	8	4,000	2,400	2,500
TW-3	1,074.0	Foam	20,000	10	25,000	1,400	1,500
TW-4	1,065.0	do	12,200	10	12,520	1,500	1,800
TW-5	1,145.0	do	53,000	2 - 6	NAP	1,900	2,000
DHM-5	1,383.5	do	40,866	7	NAP	850	950
DHM-6	1,238.6	do	50,568	3 - 7	10,000	850	900
PITTSBURGH COALBED, VESTA NO. 5 MINE, WASHINGTON COUNTY, PA							
USBM-4	588.0	Gelled water	7,300	10.5	3,500	1,550	1,800
PITTSBURGH COALBED, EMERALD MINE, GREENE COUNTY, PA							
EM-5	764.0	Foam	31,500	11.8	10,000	1,375	NR
EM-6	582.0	do	29,200	17.7	14,000	1,500	² 1,150
EM-7	728.0	do	29,000	10.8	7,400	1,400	² NR
EM-8	646.0	do	42,000	10.8	12,800	1,050	1,200
EM-11	713.0	Kiel water	54,600	19	10,000	1,200	NR
PITTSBURGH COALBED, CUMBERLAND MINE, GREENE COUNTY, PA							
CNG-1034	754.0	Foam	21,840	10	23,500	1,200	NR
UPPER FREEPORT COALBED, LUCERNE NO. 6 MINE, INDIANA COUNTY, PA							
RP-1	626.5	Water	1,600	0.4 - 7.4	NAP	NAP	1,260
RP-2	630.4	Foam	19,800	8	7,000	1,150	1,250
RP-3	634.6	do	30,200	8	11,500	850	1,110
LOWER KITTANNING COALBED, KITT NO. 1 MINE, BARBOUR COUNTY, WV							
KE-2	660.0	Foam	23,500	8	3,300	NAP	² 2,000
DGBH-5	637.0	do	50,400	8	11,800	950	1,210
ILLINOIS NO. 6 COALBED, INLAND STEEL MINE, JEFFERSON COUNTY, IL							
I-NE	729.0	Gelled water	12,000	10	6,400	850	1,050
JAWBONE COALBED, McCLURE NO. 1 MINE, DICKENSON COUNTY, VA							
DG-1A	425.0	Foam	35,300	16	28,000	NR	NR
ROCK CANYON COALBED, SOLDIER CANYON MINE, CARBON COUNTY, UT							
SC-1	684.0	Foam	50,000	16	15,000	1,350	1,500
¹ Plus 900-gal water pad							
² Screenout occurred							
³ Plus 640-gal water flush							
⁴ Plus 4,200-gal water pad							
NAP Not applicable.							
NR Not reported.							

NAP Not applicable.
NR Not reported.

¹ Plus 900-gal water pad
² Screenout occurred.

³ Plus 640-gal water flush.
⁴ Plus 4,200-gal water pad.

Table 11.—Summary of underground observations for 22 intercepted stimulation treatments

Borehole	Hole location exposed	Fluorescent paint used	Max observed vertical wing length, ft		Max width vertical fracture, in	Zones of multiple vertical fractures	Max observed horizontal fracture length, ft		Multiple horizontal fracture planes	Roof penetration
			Sand filled	Paint coated			Sand filled	Paint coated		
TW-1	Yes	No	5	NAP	1/8	Yes	NAP	NAP	NAP	Yes
TW-2	Yes	No	16	NAP	4 1/2	Yes	8	NAP	Yes	No
TW-3	Yes	No	210	NAP	1/8	No	NAP	NAP	NAP	Yes
TW-4	Yes	No	220	NAP	1/8	No	NAP	NAP	NAP	Yes
TW-5	Yes	Yes	NAP	370	Clear	Yes	NAP	230	Yes	Yes
DHM-5	No	Yes	NAP	340	Clear	Yes	NAP	NAP	NAP	Yes
DHM-6	Yes	Yes	95	630	1/8	Yes	NAP	NAP	NAP	Yes
USBM-4	Yes	No	20	NAP	2 1/2	Yes	NAP	NAP	NAP	No
EM-5	No	No	140	NAP	1 1/2	No	50	NAP	No	No
EM-6	Yes	No	10	NAP	2 1/2	Yes	10	NAP	Yes	Yes
EM-7	Yes	No	18	NAP	1/2	No	NAP	NAP	NAP	No
EM-8	No	No	55	NAP	3/8	Yes	115	NAP	Yes	Yes
EM-11	No	No	NAP	NAP	NAP	NAP	110	NAP	No	Yes
CNG-1034	No	No	100	NAP	1/8	No	NAP	NAP	NAP	No
RP-1	Yes	Yes	2	2	1/8	Yes	NAP	35	No	No
RP-2	Yes	Yes	30	67	1/8	Yes	17	265	Yes	No
RP-3	Yes	Yes	20	130	1	Yes	200	200	Yes	No
KE-2	No	Yes	NAP	85	Clear	No	NAP	100	No	No
DGBH-5	No	No	72	NAP	1/8	Yes	105	NAP	No	No
I-NE	No	No	416	NAP	1/8	No	NAP	NAP	NAP	Yes
DG-1A	Yes	No	20	NAP	1/8	NR	250	NAP	Yes	No
SC-1	No	Yes	NAP	NAP	NAP	NAP	NAP	NAP	NAP	NAP

NAP Not applicable NR Not reported ¹Sand from fill at bottom of hole, not from treatment.

commonly do not extend the entire height of the coalbed, but are found in only part of the coalbed. In the Blue Creek Coalbed, when fractures did not extend the entire height of the coalbed they were usually present in the upper part, and less commonly, only in the lower part of the coalbed. This may be related to variations in the physical character of the Blue Creek Coalbed, which is commonly more friable and "soft" in the upper part.

It has been suggested (9, 28-29) that gelled water stimulation treatments with sand proppant would theoretically produce relatively short, wide, sand-filled vertical fractures and foam treatments would produce longer, narrow, sand-filled fractures. Only four of the treatments observed underground used gelled water as the treatment fluid and therefore a definitive statement relating the relative differences between the character of fractures

resulting from the different fluids is not possible. However, the widest sand-filled fracture observed ($4\frac{1}{2}$ in, borehole TW-2) was from a gelled water treatment. A $2\frac{1}{2}$ -in-wide sand-filled fracture was also observed for the gelled water treatment on borehole USBM-4, but a $2\frac{1}{2}$ -in-wide sand-filled fracture was also observed for the foam treatment on borehole EM-6. The longest sand-filled vertical fracture observed (416 ft, borehole 1-NE) was also from a gelled water treatment where short fractures would be expected.

The type of stimulation fluid has some influence on the character of the induced fractures, as do other factors such as treatment volume, injection rate, and depth and physical characteristics of the coalbed. Larger volume treatments tended to have more observable fractures as was seen in the three progressively larger treatments in the Upper Freeport Coalbed (boreholes RP-1, RP-2, and RP-3). Boreholes TW-5 and DHM-6, which had two of the largest volume treatments also had some of the most extensive networks of fractures.

The physical properties of the coalbed seemed to have some influence on the propagation of fractures. Many of the vertical fractures in the Blue Creek Coalbed were present only in the more friable upper part. The vertical sand-filled fractures at borehole TW-2 were located in the bottom "hard" part of the Blue Creek Coalbed and inclined and became horizontal at or near the interface between the lower "hard" part and the upper "soft" friable part. The Upper Freeport Coalbed (boreholes RP-1, RP-2, and RP-3) had a distinct shale parting near the top of the coalbed. In some cases the parting acted as a barrier to the upward growth of fractures, but it was more commonly penetrated by the vertical fractures, which then became horizontal at the interface at the top of the parting and base of the upper coal bench. A distinct "shaley" band in the Pittsburgh Coalbed at borehole EM-8 was generally penetrated by the vertical fractures that continued vertical above, but in one instance the fracture became horizontal above the band. It was also common to see small horizontal offsets in the vertical propagation of the fractures as different layers or bedding planes in the coalbed were encountered. These offsets occasionally resulted in an inclined or sinuous appearance to the "vertical" fracture. While the physical properties of the coalbed seemed to have some influence on the character of the fractures, that influence was variable and could not be considered predictable.

A horizontal fracture was discernable in approximately half of the mined through stimulation treatments. Five of eight treatments that used fluorescent paint and seven additional treatments that did not use paint had identifiable horizontal fractures. It is generally thought that at increasing depths (overburden pressure) the incidence of horizontal fracturing decreases. A horizontal fracture was found in stimulated coalbeds as deep as 1,145 ft in the Blue Creek Coalbed (borehole TW-5). This fracture was only identifiable as paint coatings since no sand was used in the treatment on borehole TW-5. The two treatments with fluorescent paint in the Blue Creek Coalbed that did not have identifiable horizontal fractures were, at depths of 1,384 ft (borehole DHM-5) and 1,239 ft (borehole DHM-6), the deepest stimulations mined through. The deepest stimulation with a sand-filled horizontal fracture was 1,093 ft in the Blue Creek Coalbed at borehole TW-2, however, the rest of the sand-filled horizontal fractures were found at depths of 400 to 800 ft.

It seems likely that additional horizontal fractures (or fluid pathways) would have been observed, especially in the shallow coalbeds, if fluorescent paint had been used in more treatments.

The most common location for the horizontal fractures is at the top of the coalbed at the interface with the roof rock. Occasionally horizontal fractures are found along other distinct interfaces such as shale partings, shaley or hard, dense bands, or rider coals in the immediate roof rock. The distribution of sand or paint on horizontal surfaces was generally erratic (partially because of the difficulty in mapping their entire area), but in two cases (boreholes RP-2 and RP-3) there was some suggestion that there was a correlation between cleat orientation and horizontal lobes of paint. Also at borehole TW-5 the horizontal (and vertical) fractures covered an area with a general elliptical shape with the long axis parallel to the face cleat orientation. Multiple horizontal fractures on different planes have been observed in several cases.

The sand-filled horizontal fractures are generally thickest near the wellbore (maximum of 1 in), usually are thin (less than $\frac{1}{2}$ in) but quite variable in thickness along short lateral distances, and may be discontinuous along an exposed rib. The maximum lateral extents for sand-filled horizontal fractures were 250 ft in the Jawbone Coalbed (borehole DG-1A) and 200 ft in the Upper Freeport Coalbed (borehole RP-3). The maximum lateral extents for paint-coated horizontal fractures were 265 and 200 ft in the Upper Freeport Coalbed (boreholes RP-2 and RP-3) and 230 ft in the Blue Creek Coalbed (borehole TW-5).

Penetration by fluids and/or sand proppant into strata directly overlying the main bench of coal has been observed in nearly half of the treatments intercepted underground. In addressing the significance of roof penetration it is necessary to point out that roof penetration does not necessarily equate to roof damage or an adverse effect on mining. There have been no roof falls attributed to the penetration of stimulation fluids into roof strata from Government-sponsored treatments. Lambert (9) did report that supplementary roof support was installed as a precautionary measure in the vicinity of borehole TW-3 at the Oak Grove Mine, where both the mined Blue Creek Coalbed and the overlying Mary Lee Coalbed, $5\frac{1}{2}$ ft above, were stimulated and where mine management observed "roof movement" along a nearby rib. According to Lambert (9), the area was mined through "without experiencing any roof fall during mining operations."

Except for the Blue Creek Coalbed, most of the penetrations of strata above the main coal bench have been fairly limited in vertical and/or horizontal extent. In several cases, penetration of strata above the Blue Creek Coalbed was observed at locations a hundred feet or more from a borehole. The reason for the more extensive roof penetrations is not conclusively known, but may be related to the complex structural history of the area. Recent *in situ* state of stress (ISSOS) tests conducted near the Oak Grove Mine indicated lower *in situ* stress values for the rocks surrounding the Mary Lee-Blue Creek Coalbeds than measured in the coal (30). In the absence of a stress barrier or mechanical property barrier, upward fracture breakout is more likely. Upward fracture breakout from this coal section was reported during the ISSOS testing near the Oak Grove Mine. The presence of

naturally occurring roof joints coupled with lower or similar in situ stresses above the coal probably influenced the extent of roof penetration at the Oak Grove Mine.

In several cases the strata penetrated above coalbeds have been weak, thin shales below rider coals, and the fractures have been contained in strata that was generally mined along with the main bench of coal throughout the mine to prevent it from deteriorating and falling at a later time. Thus such strata penetration should not really be considered roof penetration.

Part of the problem of acceptance of the use of stimulated vertical boreholes to remove gas from coalbeds prior to mining may be related to the use of the terms

"fracture," "fracturing," and "breakdown," which perhaps suggest a catastrophic breakup of the strata treated. The evidence from direct underground observation and many of the treatment records (including those from boreholes not mined through) suggest that new fractures are seldom created, but rather naturally occurring planes of weakness (cleat, joints, or bed boundaries) are entered and widened to varying degrees. In most cases the penetration of strata overlying the main coal bench has been attributed to the invasion of preexisting joints, as evidenced by similar joints of the same general character and orientation occurring throughout a mine.

CONCLUSIONS

It is impossible to guarantee that a stimulation treatment in a coalbed will not adversely affect mining in some way. However, the underground evidence and experience to date suggests that the probability of adversely affecting mining conditions is minimal. The use

of prestimulation strata characterization tests and informed treatment design and controlled implementation (primarily injection rates and therefore treatment pressure) can probably further minimize the chance of adverse mining conditions.

REFERENCES

1. Finfinger, G. L., and J. Cervik. Review of Horizontal Drilling Technology for Methane Drainage From U.S. Coalbeds. BuMines IC 8829, 1980, 20 pp.
2. Hagood, D. W., R. C. Pate, and J. W. Stevenson. Methane Control in an Advancing Section of an Underground Coal Mine (contract S0395033, Jim Walters Resour. Inc.). BuMines OFR 94-84, 1983, 39 pp.; NTIS PB 84-185057.
3. Perry, J. H., G. N. Aul, and J. Cervik. Methane Drainage Study in the Sunnyside Coalbed, Utah. BuMines RI 8323, 1978, 11 pp.
4. Prosser, L. J., Jr., G. L. Finfinger, and J. Cervik. Methane Drainage Study Using an Underground Pipeline, Marianna Mine 58. BuMines RI 8577, 1981, 29 pp.
5. Thakur, P. C., and W. N. Poundstone. Horizontal Drilling Technology for Advance Degasification. Min. Eng. (Littleton, CO), v. 32, No. 6, 1980, pp. 676-680.
6. Von Schonfeldt, H., D. R. Pothini, G. N. Aul, and R. L. Henderson. Production and Utilization of Coalbed Methane Gas in Island Creek Coal Company Mines. Paper in Proceedings of the Unconventional Gas Recovery Symposium (May 16-18, 1982, Pittsburgh, PA). SPE DOE 18817, 1982, pp. 235-244.
7. Dunn, B. W. Coal as a Conventional Source of Methane: A Review and Analysis of 50 Wells in Two Production Areas in the Black Warrior Basin of Alabama. Paper in Proceedings of the Unconventional Gas Recovery Symposium (May 13-15, 1984, Pittsburgh, PA). SPE DOE GRI 12875, 1984, pp. 407-416.
8. Elder, C. H. Effects of Hydraulic Stimulation on Coalbeds and Associated Strata. BuMines RI 8260, 1977, 20 pp.
9. Lambert, S. W., M. A. Trevita, and P. F. Steidl. Vertical Borehole Design and Completion Practices To Remove Methane Gas From Mineable Coalbeds. U.S. Dep. Energy CMTCTR-80-2, 1980, 163 pp.
10. Steidl, P. F. Foam Stimulation To Enhance Production From Degasification Wells in the Pittsburgh Coalbed. BuMines RI 8286, 1978, 10 pp.
11. Stubbs, P. B., F. X. Dobscha, and J. V. Mahoney. Degasification of the Blue Creek Coal Seam at Oak Grove Mine. Paper in Proceedings of the 2d Annual Symposium on Methane-Recovery-From Coalbeds (Apr. 18-20, 1979, Pittsburgh, PA) ed. by R. L. Wise. U.S. Dep. Energy, METC SP-79-9, 1979, pp. 96-113.
12. Murrie, G. W., J. T. Popp, C. M. McCulloch, and D. J. Steele. Development and Evaluation of Near Mine Stimulation Techniques for Coalbed Degasification (contract J0333908, Dames and Moore). BuMines OFR 135-84, 1984, 127 pp.; NTIS PB 84-220664.
13. McCulloch, C. M., S. W. Lambert, and J. R. White. Determining Cleat Orientation of Deeper Coalbeds From Overlying Coals. BuMines RI 8116, 1976, 19 pp.
14. Diamond, W. P., C. M. McCulloch, and B. M. Bench. Use of Surface Joint and Photolinear Data for Predicting Subsurface Coal Cleat Orientation. BuMines RI 8120, 1976, 13 pp.
15. Diamond, W. P., G. W. Murrie, and C. M. McCulloch. Methane Gas Content of the Mary Lee Group of Coalbeds, Jefferson, Tuscaloosa, and Walker Counties, Ala. BuMines RI 8117, 1976, 9 pp.
16. Murrie, G. W., W. P. Diamond, and S. W. Lambert. Geology of the Mary Lee Group of Coalbeds, Black Warrior Basin, Alabama. BuMines RI 8189, 1976, 49 pp.
17. Boyer, C. M., and P. B. Stubbs. Measurement of Dynamic Coalbed Reservoir Conditions Required To Design Near-Mine Methane Control Activities in the Warrior Coal Basin. U.S. Dep. Energy DOE ET-14204-1477, Feb. 1983, 133 pp.
18. Lambert, S. W. (Taurus Exploration, Inc.). Private communication, 1985; available upon request from S. W. Lambert, Taurus Exploration, Inc., Tuscaloosa, AL.
19. Boyer, C. M. (U.S. Steel Corp.). Private communication, 1985; available upon request from C. M. Boyer, U.S. Steel Corp., Monroeville, PA.
20. Lambert, S. W., and M. A. Trevita. Methane Drainage: Experience With Hydraulic Stimulation Through Slotted Casing. BuMines RI 8295, 1978, 16 pp.
21. ———. Methane Drainage Ahead of Mining Using Foam Stimulation-Mary Lee Coalbed, AL. U.S. Dep. Energy RI-PMTC-3(79), Jan. 1979, 22 pp.
22. Mahoney, J. V., P. B. Stubbs, F. C. Schwerer, and F. X. Dobscha. Effects of a No-Proppant Foam Stimulation Treatment on a Coal Seam Degasification Borehole. J. Pet. Technol., v. 33, No. 11, 1981, pp. 2227-2235.
23. Kiel, O. M. Hydraulic Fracturing Process Using Reverse Flow. U.S. Pat. 3933205, Jan. 20, 1976.

24. Frantz, R. L., J. F. Keeney, and R. Stefanko. Feasibility Studies and Demonstration of Methane Recovery From Pennsylvania Coal Seams (contract J0155168, PA State Univ.). BuMines OFR 4-79, 1978, 187 pp.
25. Towae, D. F., M. E. Hanson, M. S. Costantino, A. Kusubov, B. J. Qualheim, and L. D. Thorsen. Coalbed Methane Research Annual Report—FY 82. U.S. Dep. Energy UCRL-53421, Mar. 1983, 36 pp.
26. Engelder, T. Is there a Genetic Relationship Between Selected Regional Joints and Temporary Stress Within the Lithosphere of North America? *Tectonics*, v. 1, No. 2, Apr. 1982, pp. 161-177.
27. Blevins, T. C. Coping With High Lateral Stresses in an Underground Illinois Coal Mine. Soc. Min. Eng. AIME preprint 82-156, 1982, 7 pp.
28. Halliburton Services. The Fracbook Design Manual for Hydraulic Fracturing. Halliburton Services, Duncan, OK, 1971, 111 pp.
29. Perkins, T. K., and L. R. Kern. Widths of Hydraulic Fractures, Paper in Proceedings of the 36th Annual Fall Meeting (Soc. Petrol. Engr., AIME, Oct. 8-11, 1961). SPE-AIME, 1961, pp. 265-277; *J. Pet. Technol.*, v. 13, No. 9, 1961, pp. 937-949.
30. Gas Research Institute Multiple Coal Seam Project. Art. in Quarterly Review of Methane From Coal Seams Technology. V. 3, No. 2, Sept. 1985, pp. 39-48.

APPENDIX.—ACQUISITION AND LIMITATIONS OF STIMULATION TREATMENT DATA

There are large measurement uncertainties associated with the recording of any hydraulic stimulation treatment. The reader should keep this fact firmly in mind when reading the treatment records reproduced in this report. In most cases where two or more independent records of treatment parameters were kept, these records seldom agreed, although supposedly all of the information in the records came from the same instruments at the same time. The records available from the treatments include the recorded pressure and water flow data (and any other parameters that may have been recorded continuously), the notes usually made by the engineer in charge of the treatment, and in some cases notes made by Government or other personnel during the treatment.

Two measurements commonly are recorded during every hydraulic stimulation treatment. These are the water pumping rate at the pump truck and the surface pressure. In addition, during some treatments, additional information can be, but typically is not, recorded. This additional information includes the density of the fracturing fluid at the pump truck, the density of the fluid as it goes into the wellhead, the bottom hole pressure, and the fracturing fluid flow rate at the wellhead.

The liquid pumping rates are determined by counting pump strokes and multiplying by a factor based upon the cylinder volume and the pump efficiency. The accuracy of the measurements is probably about 5 pct. Nitrogen for foam fracturing treatments is carried as a liquid, so liquid flowmeters are used to measure the nitrogen flow rate before it is heated and vaporized. The accuracy of the meters is not commonly given, but is probably no better than 5 pct. Surface pressures may be measured using a variety of types of instruments. Pressure should be the most accurately determined quantity of any measured during a treatment. It should be possible to determine treating pressures to an accuracy of 1 pct (or within 10 lbf/in² per 1,000 lbf/in² ga). In practice an accuracy of ± 100 lbf/in² is probably very good.

The fluid density is determined by using a densitometer employing a gamma ray source and a detector that counts the gamma rays returning through the fluid. Most pump trucks carry one to allow measurement and control of cement density while cementing casing in oil and gas wells. The instruments are calibrated for higher densities than those of fracturing fluids, but they should still be usable, and they were used to measure sand concentrations during the treatments of some of the wells described in this report.

The exact method of measuring the sand injection rate is unclear, but probably it is estimated from the density of the sand and water mixture at the truck. The method of determining the total weight of sand injected at the end of the treatment appears to be to take the difference between the original sand volume carried to the site and the remaining sand volume after completion of the treatment. If this is the case, then the sand injection rates given during the course of the treatments (and shown on some of the treatment records in this report) are estimates only and do not necessarily give accurate sand injection rates.

Bottom hole pressure may be measured in two different ways. The most accurate method, but for practical reasons the most difficult and expensive to

employ, is to place a remote-reading pressure gauge at the bottom of the hole. A simpler method is to run tubing in the hole to the depth of the coalbed and fill the tubing with water. At the wellhead a pressure gauge is then installed on the top of the tubing. The pressure gauge on top of the tubing then reads the bottom hole pressure less the hydrostatic pressure of the water in the tubing. The hydrostatic pressure can then be added back in to obtain the bottom hole pressure. The treatment is conducted by pumping fluids down the annulus between the casing and the tubing. The method suffers from the drawback that the tubing can be sealed off by coal or sand, or gas can enter the tubing, causing incorrect pressure readings. This method was used in the wells described in this report where bottom hole pressures are given. It should be kept in mind that bottom hole pressures can differ from surface pressures because of the hydrostatic pressure of the treating fluid in the borehole and because of friction losses due to pumping. These two sources of pressure differences have opposite effects, so the bottom hole pressure can be less than, greater than, or the same as the surface pressure, depending upon the conditions of the treatment.

During gelled water treatments, all measurements at the wellhead will be the same as those at the truck and are redundant. However when foam is being pumped, it is necessary to determine the density and flow rate of both the water and sand mixture leaving the pump truck and of the foam mixture after the nitrogen has been added (downstream of the pump truck) in order to determine the true volumetric pumping rate of the foam (a water, nitrogen, and sand mixture). This additional information has been obtained for some of the wells described in this report, but in general is not available for most foam-stimulated wells.

Because the foam mixture is a compressible fluid, its actual (volumetric) pumping rate is controlled by the pressure. During a typical foam stimulation treatment the liquid and gas flow rates are measured separately and the volumes are then added, with the nitrogen flow rate being determined at the surface pressure. An attempt is generally made to maintain a foam quality of 75 pct (or 75 pct nitrogen by volume and 25 pct sand and water by volume) at the surface for optimum sand carrying capacity. This is done by estimating the average treating pressure at the beginning of the treatment and calculating the nitrogen required to give a gas volume of 75 pct of the total foam volume.

Occasionally during a treatment as pressures change, the nitrogen rate is corrected, but difficulties in adjusting the rate of the nitrogen flow make service company personnel reluctant to continually adjust the nitrogen flow rate to maintain the proper foam quality. Because of this, large errors in the estimated volume of the treatment can occur. The estimated treating rate is obtained by multiplying the water rate by a factor of 4. This is only correct as long as the foam quality is 75 pct. However if the pressure deviates from the initial planned treating pressure (or indeed if the estimate was incorrect to begin with) then the foam quality may never be 75 pct.

Foam qualities as low as 50 pct have been observed and 60 pct is common. If a treatment planned for a 75-pct foam quality is actually run at 50-pct quality, then the true foam volume pumped will only be one half of that

initially planned. A planned 50,000-gal foam treatment could then become a 25,000-gal treatment with the same material masses pumped into the formation. In addition, all of these measurements are made at surface conditions and the foam quality is adjusted for surface conditions. However it is the bottom hole foam quality that really should be maintained at 75 pct and without knowing the bottom hole pressure there is no guarantee that the bottom hole foam quality is correct.

As an example, consider a treatment planned for a foam injection rate of 10 bbl/min at a pressure of 500 lbf/in²(abs). The required nitrogen rate to maintain a 75-pct quality at this pressure is 185 std ft³/min. If the pressure increases to 1,000 lbf/in²(abs) during the treatment, the required nitrogen volume is doubled to 370 std ft³/min. If the nitrogen rate is not increased, which is commonly the case, then the foam quality is reduced to 50 pct and the actual pumping rate becomes 5 bbl/min. This effect is more pronounced at the lower pressures used in shallow coalbed wells than in deeper holes, which require higher treating pressures. For instance it would require a pressure increase from 1,000 to 2,100 lbf/in²(abs) to necessitate another doubling of the nitrogen rate. Pressure changes of 500 lbf/in² are common during the course of a foam stimulation, but changes of 1,000 lbf/in² occur much less frequently.

The fact that the measurement errors can be much larger at low pressures is a problem because the service company personnel, being familiar with oil and gas wells treated at much higher pressures are less likely to consider it necessary to adjust nitrogen rates during the course of a treatment. Their reluctance to adjust the nitrogen rate is also increased because it is difficult to adjust the nitrogen rate and because the very act of increasing the nitrogen rate leads to changes in volumetric flow rate, which can increase the pressure leading to a need for further increases in nitrogen flow. It may take some time to reach an equilibrium flow rate. Starting with too much nitrogen is not the answer because at high qualities (possibly 85 pct and certainly 90 pct) the foam begins to lose its sand carrying properties.

The measurements of pumping rates, pressures, densities, and sand weights are not always highly accurate. In the few cases observed by the Bureau where two or more independent measurements of the same quantity were available, the measurements disagreed by a large amount. In the case of a treatment at the Soldier Canyon Mine, Utah, at one point there were three surface pressure readings available ranging from about 1,500 to

2,000 lbf/in²(ga). None of the three gauges agreed and two showed differences of 300 lbf/in²(ga) (a 20-pct difference). Pressure should be the easiest treatment parameter to measure accurately. Pumping rates, densities, and nitrogen rates are more difficult to measure accurately and it is therefore reasonable to have even greater suspicion of treatment volumes computed from measurements of these parameters.

One way to potentially improve the reliability of the treatment records is for customers to discuss the planned treatment and the measuring techniques in detail with the service company sales engineers well in advance of the treatment and with the field engineer just prior to the treatment. The actual effectiveness of this scenario is probably limited, however. The most important individual to discuss the treatment with prior to its start is usually the busiest and most difficult to arrange time to talk to. In addition, this individual may in some cases misinterpret an interest in the details of a treatment as suggesting a lack of trust on the customer's part. Requests for information on the accuracy of gauges and meters may be met with comments that the measurements are exactly correct, and that there are no measurement errors, or the engineer may simply not know the accuracy of his or her equipment. The same is true of a request for information on the exact method of operation of the equipment and the mode of measurement of the treatment parameters. These parameters can vary with the equipment used and can only be determined with certainty at the job site by discussion with the field engineers.

One important area where improvement is perhaps more likely is in recordkeeping. On many of the records studied for this report it was found that service company personnel tended to leave out scale information on the pressure and flow rate charts. The scale information most often left off was the time scale. Customers should be careful to request that this information be placed on these records or to note the scales themselves and add them to the records when they obtain them. It should also be made clear to the service company personnel by the customers that they want copies of these records, since copies are always made, but not necessarily given to the customer unless requested. It is also important that the customers keep their own notes as a check on the accuracy of the service company field engineer's field notes. The job ticket used for computing the treatment cost should not be trusted to give an accurate history of the treatment, since this record may be altered in the field for various reasons, such as giving unadvertised job discounts.